**ENERGY, MINES AND RESOURCES CANADA**

**CANADA CENTRE FOR MINERAL AND ENERGY TECHNOLOGY**

**NATIONAL URANIUM TAILINGS PROGRAM**

**FINAL REPORT**
**FOR**

# COST STUDY ON WASTE MANAGEMENT AT THREE MODEL CANADIAN URANIUM MINES

**MARCH, 1984**

**KILBORN** (SASKATCHEWAN) LTD.
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April 3, 1984

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Mines and Resources
Canada Centre for Mineral
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ATTENTION: MR. D. G. FEASBY

Dear Mr. Feasby:

We are pleased to submit twenty (20) copies of the final report entitled "Cost Study on Waste Management at Three Model Canadian Uranium Mines" which was completed for the National Uranium Tailings Program under contract to the Department of Supply and Services.

We trust the information included in this report meets your requirements and we remain available to answer any questions or discuss any aspects of the report at your convenience.

We wish to express our sincere appreciation for the opportunity to be of service to the National Uranium Tailings Program. We look forward to providing further assistance in the future.

Respectfully submitted,

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REQ. NO. 3-1653

CONTRACT REPORT NO. OSQ83-00232

# COST STUDY ON WASTE MANAGEMENT
# AT
# THREE MODEL CANADIAN URANIUM MINES

prepared for

DEPARTMENT OF ENERGY, MINES AND RESOURCES
CANADA CENTRE FOR MINERAL AND ENERGY TECHNOLOGY
NATIONAL URANIUM TAILINGS PROGRAM

by

KILBORN (SASKATCHEWAN) LTD.
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247 pp.

JUL 30 1984
MICROMEDIA

March, 1984

Saskatoon, Canada

# COST STUDY ON WASTE MANAGEMENT

# AT

# THREE MODEL CANADIAN URANIUM MINES

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# COST STUDY ON WASTE MANAGEMENT
# AT
# THREE MODEL CANADIAN URANIUM MINES

## 1.0 EXECUTIVE SUMMARY

### INTRODUCTION

As part of the National Uranium Tailings Program, Kilborn (Saskatchewan) Ltd. in association with Melis Consulting Engineers Ltd., and with the assistance of Knight and Piesold Ltd., were commissioned to carry out a waste management cost study under contract to the Department of Supply and Services. This cost study was initiated to determine the capital and operating costs of three different uranium waste management systems which incorporate current technologies being used in Canadian uranium mining operations. Cost estimates were to be done to a thirty percent level of accuracy and were to include all waste management related costs of a uranium ore processing facility.

Each model is based on an annual uranium production of 1,923,000 kg U (5,000,000 lbs $U_3O_8$) with a total operating life of 20 years for the facility. The three models, A, B, and C, are based on three different uranium ore grades, 0.10% $U_3O_8$, 0.475% $U_3O_8$ and 1.5% $U_3O_8$ respectively. Yellowcake production is assumed to start in January 1984.

### PROCESS DESCRIPTION

Model A is based on a conceptual 7,180 tonne per day uranium ore processing facility and waste management system typical of uranium operations in the Elliot Lake area of northern Ontario with an established infrastructure.

## 1.0 (Cont'd)

Bulk neutralization of mill tailings is practiced with tailings decant water being treated for acid neutralization, radium removal and clarification in settling ponds. Acidic minewater from underground mining operations is neutralized separately for the recovery of the contained uranium with the resulting neutralized minewater being used as process water. Tailings containment is by a conventional sub-aqueous system in a natural lake basin using open-end discharge of the tailings slurry.

Model B is a 1,512 tonne per day operation based on a remote uranium operation typical of the Athabasca Basin properties in northern Saskatchewan. Model B processes a relatively clean uranium ore utilizing conventional sub-aqueous tailings containment in a tailings basin constructed of engineered dykes. Bulk tailings neutralization is also practiced in this model with the tailings decant water being treated for radium removal followed by clarification with sand filtration. Minewater from open-pit and underground operations is clarified and collected in a clarification pond prior to use as process water.

Model C is a 466 tonne per day operation processing a high-grade uranium ore containing arsenic and heavy metal concentrations typical of some northern Saskatchewan deposits. Sub-aerial tailings deposition in a tailings basin with an extensive underdrainage system is used in this model. Seepage and runoff from the tailings basin are recycled to the mill for treatment in the mill effluent treatment circuit. Waste sludge from the effluent treatment circuit is combined with neutralized mill tailings for disposal to the tailings area.

## 1.0 (Cont'd)

Contaminated minewater from the open-pit operation in Model C is clarified in a clarification pond and used as process water with any excess water being treated in a separate treatment circuit. The waste precipitates from the minewater treatment circuit are collected in settling ponds and the waste sludge is transferred to the tailings basin on an annual basis.

Treated effluent is clarified with sand filters and discharged to monitoring ponds for check-analysis prior to discharge to the environment. The uranium recovery process for this model established the need for an ammonium sulphate crystallization plant for the control of ammonia in the treated effluent discharging to the local watershed.

### COST ESTIMATES

The cost estimates for each model are based on the design basis provided in the original Statement of Work (the terms of reference for this study) and on operating information obtained from site visits to similar uranium mining operations. All costs are in first quarter 1984 Canadian dollars.

The capital costs include all waste management related costs of a uranium ore processing facility including minewater treatment, mill effluent and tailings treatment, required reagent facilities, tailings lines, tailings storage basin, tailings decant treatment, monitoring requirements and associated buildings. As well as direct costs, the capital cost estimate includes construction indirects, capital spares allowance, engineering, procurement and construction management, and a contingency allowance. A sustaining capital cost is included for the Model B tailings basin in order to meet the requirement of staged construction of the tailings basin dykes as assumed in the study.

## 1.0 (Cont'd)

The annual operating costs for each model include all operating requirements of the waste management section of each facility. This includes supervision and hourly labour, reagents and supplies, power, heating, road and yard maintenance, monitoring requirements and a contingency allowance. Only the costs directly associated with waste management for each facility have been included. Models B and C also include an allowance for company-sponsored transportation and accommodation to account for the additional cost of the 7-in, 7-out work schedule typical of northern Saskatchewan operations.

A summary of capital and operating costs of the waste management system for each model is given in Table 1.1 along with the basic production assumptions for each model.

The total capital cost is a summation of pre-production capital, working capital and sustaining capital in constant 1984 dollars. The pre-production capital includes the capital cost of the facility prior to start-up. Working capital is an allowance for the initial charge of consumables in the process. Sustaining capital is an allowance for capital requirements during the life of the operation. The sustaining capital is considered negligible in Models A and C but a total of $20,200,000 is required for Model B to allow for the staged construction of the tailings basin dykes as assumed in the study.

The annual operating costs for each model, in first quarter 1984 dollars, include all the associated costs of operating the waste management facility of each model. Unit operating costs are presented in terms of dollars per tonne of ore processed ($/tonne), dollars per kilogram U produced ($/kg U) and dollars per pound $U_3O_8$ produced ($/lb $U_3O_8$).

## 1.0 (Cont'd)

The total costs of the waste management system of each model for a 20-year operation are also presented in Table 1.1. This total cost, in constant 1984 dollars, includes pre-production capital, initial working capital, sustaining capital and the total operating costs over the 20-year life of the operation.

Unit total costs in Table 1.1 are expressed in terms of dollars per tonne based on the total ore throughput of each model during the 20-year operation as well as dollars per kilogram U and dollars per pound $U_3O_8$ based on a total uranium production of 38,465,000 kg U (100,000,000 lbs $U_3O_8$) over a 20-year operating life.

A before-tax net present value (NPV), in constant 1984 dollars, is given for each model in Table 1.1. This calculation is based on the cost elements of the cash flow for each model given in Tables 4.8, 5.9 and 6.8 of this report, using a discount rate of 12.5%. It should be noted that no attempt has been made to account for the different tax and royalty structures that exist between provinces for uranium operations. Since this calculation is based on a pre-tax basis, there has also been no consideration given for capital cost allowance or depreciation.

A more detailed presentation of the cost estimates has been accomplished by proportioning the capital and operating costs of each model to the following areas of waste management within each model:

- minewater treatment
- mill effluent treatment
- tailings line
- tailings storage basin
- tailings decant treatment
- monitoring

## TABLE 1.1

## SUMMARY OF TOTAL COSTS OF WASTE MANAGEMENT AT THREE MODEL CANADIAN URANIUM MINES

(FIRST QUARTER 1984 DOLLARS)

| | MODEL A | MODEL B | MODEL C |
|---|---|---|---|
| Location | Elliot Lake, Ontario | Northern Saskatchewan | Northern Saskatchewan |
| Ore Grade - % U | 0.085 | 0.40 | 1.27 |
| - % $U_3O_8$ | 0.10 | 0.475 | 1.5 |
| Milling Capacity (tonnes/day) | 7180 | 1512 | 466 |
| Operating life (years) | 20 | 20 | 20 |
| Annual Production - kg U | 1,923,000 | 1,923,000 | 1,923,000 |
| - lbs $U_3O_8$ | 5,000,000 | 5,000,000 | 5,000,000 |
| Capital Costs | | | |
| Pre-production ($) | 30,663,400 | 25,445,000 | 47,665,000 |
| Working Capital ($) | 300,000 | 300,000 | 500,000 |
| Sustaining Capital ($) | - | 20,200,000 | - |
| Total Capital ($) | 30,963,400 | 45,945,000 | 48,165,000 |
| Operating Costs | | | |
| Annual ($) | 3,982,200 | 4,357,400 | 8,579,700 |
| $/tonne processed | 1.67 | 8.67 | 55.32 |
| $/kg U produced | 2.07 | 2.27 | 4.46 |
| $/lb $U_3O_8$ produced | 0.80 | 0.87 | 1.72 |
| Total Costs | | | |
| Total ($) | 110,607,000 | 133,093,000 | 219,759,000 |
| $/tonne processed | 2.32 | 13.24 | 70.85 |
| $/kg U produced | 2.88 | 3.46 | 5.71 |
| $/lb $U_3O_8$ produced | 1.11 | 1.33 | 2.20 |
| Net Present Value (discount rate of 12.5%) | -44,410,000 | -46,540,000 | -81,090,000 |

## 1.0 (Cont'd)

The capital cost estimate of each major area of the waste management system of Models A, B and C are summarized in Tables 1.2, 1.3 and 1.4 respectively. Unit capital costs relative to the annual uranium production and annual tonnage throughput for each major area are also presented for each model. These unit costs are given in terms of dollars per annual tonne of ore processed ($/tonne), dollars per annual kilogram U produced ($/kg U) and dollars per annual pound $U_3O_8$ ($/lb $U_3O_8$).

It should be noted that these capital cost estimates do not include any provision for pre-production construction interest since interest costs are dependent on the method of project financing and the equity or credit position of the specific investor. These pre-production interest costs can have a significant impact on the total costs of any project and this should be kept in mind when reviewing the cost estimates given in this report.

The waste management related operating costs for each major area of Models A, B and C are summarized in Tables 1.5, 1.6 and 1.7 respectively. These costs, in first quarter 1984 dollars, include all the associated costs of operating the waste management facility for each model representing a uranium processing facility producing 1,923,000 kg U (5,000,000 lbs $U_3O_8$) on an annual basis.

Unit operating costs relative to the annual uranium production and annual tonnage throughput for each major waste management area are also given. These unit costs are presented in terms of dollars per tonne of ore processed ($/tonne), dollars per kilogram U produced ($/kg U) and dollars per pound $U_3O_8$ produced ($/lb $U_3O_8$).

## TABLE 1.2

## MODEL A
## CAPITAL COST SUMMARY

| Area | Dollars | $/annual tonne | $/annual kg U | $/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 6,810,900 | 2.85 | 3.54 | 1.36 |
| Mill Effluent Treatment | 10,867,500 | 4.55 | 5.65 | 2.17 |
| Tailings Line | 962,400 | 0.40 | 0.50 | 0.19 |
| Tailings Storage Basin | 7,707,900 | 3.23 | 4.01 | 1.54 |
| Tailings Decant Treatment | 3,968,900 | 1.66 | 2.06 | 0.81 |
| Monitoring | 345,800 | 0.15 | 0.18 | 0.07 |
| TOTAL | 30,663,400 | 12.84 | 15.94 | 6.14 |

## TABLE 1.3

## MODEL B
## CAPITAL COST SUMMARY

| Area | Dollars | \$/annual tonne | \$/annual kg U | \$/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 355,200 | 0.71 | 0.19 | 0.07 |
| Mill Effluent Treatment | 3,352,900 | 6.67 | 1.74 | 0.67 |
| Tailings Line | 411,300 | 0.82 | 0.21 | 0.08 |
| Tailings Storage Basin | 17,760,500 | 35.34 | 9.23 | 3.55 |
| Tailings Decant Treatment | 3,188,600 | 6.34 | 1.66 | 0.64 |
| Monitoring | 376,500 | 0.75 | 0.20 | 0.08 |
| Subtotal | 25,445,000 | 50.63 | 13.23 | 5.09 |
| Sustaining Capital (Tailings Basin) | 20,200,000 | 40.19 | 10.50 | 4.04 |
| TOTAL | 45,645,000 | 90.82 | 23.73 | 9.13 |

## TABLE 1.4

## MODEL C
## CAPITAL COST SUMMARY

| Area | Dollars | $/annual tonne | $/annual kg U | $/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 4,820,100 | 31.08 | 2.51 | 0.96 |
| Mill Effluent Treatment | 22,879,900 | 147.54 | 11.90 | 4.58 |
| Tailings Line | 2,308,000 | 14.88 | 1.20 | 0.46 |
| Tailings Storage Basin | 16,028,400 | 103.36 | 8.33 | 3.21 |
| Monitoring | 1,628,600 | 10.50 | 0.85 | 0.33 |
| TOTAL | 47,665,000 | 307.36 | 24.79 | 9.54 |

## TABLE 1.5

## MODEL A
## ANNUAL OPERATING COST SUMMARY

| Area | Dollars | \$/tonne processed | \$/kg U produced | \$/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Minewater Treatment | 872,880 | 0.37 | 0.46 | 0.17 |
| Mill Effluent Treatment | 2,716,557 | 1.14 | 1.41 | 0.54 |
| Tailings Line | 26,921 | 0.01 | 0.01 | 0.01 |
| Tailings Storage Basin | 40,071 | 0.02 | 0.02 | 0.01 |
| Tailings Decant Treatment | 196,478 | 0.08 | 0.10 | 0.04 |
| Monitoring | 129,281 | 0.05 | 0.07 | 0.03 |
| TOTAL | 3,982,188 | 1.67 | 2.07 | 0.80 |

## TABLE 1.6

## MODEL B
## ANNUAL OPERATING COST SUMMARY

| Area | Dollars | $/tonne processed | $/kg U produced | $/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Minewater Treatment | 65,785 | 0.13 | 0.03 | 0.01 |
| Mill Effluent Treatment | 3,340,865 | 6.65 | 1.74 | 0.67 |
| Tailings Line | 63,554 | 0.13 | 0.03 | 0.01 |
| Tailings Storage Basin | 39,600 | 0.08 | 0.02 | 0.01 |
| Tailings Decant Treatment | 488,244 | 0.97 | 0.26 | 0.10 |
| Monitoring | 359,324 | 0.71 | 0.19 | 0.07 |
| TOTAL | 4,357,372 | 8.67 | 2.27 | 0.87 |

## TABLE 1.7

## MODEL C
## ANNUAL OPERATING COST SUMMARY

| Area | Dollars | $/tonne processed | $/kg U produced | $/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Minewater Treatment | 1,581,064 | 10.19 | 0.82 | 0.32 |
| Mill Effluent Treatment | 6,193,698 | 39.94 | 3.22 | 1.24 |
| Tailings Line | 81,039 | 0.52 | 0.04 | 0.02 |
| Tailings Storage Basin | 162,309 | 1.05 | 0.09 | 0.03 |
| Monitoring | 561,557 | 3.62 | 0.29 | 0.11 |
| TOTAL | 8,579,667 | 55.32 | 4.46 | 1.72 |

**1.0 (Cont'd)**

## DISCUSSION

The cost estimates given in Tables 1.1 to 1.7 reflect the waste management costs for three different uranium mine models using current tailings disposal methods typical of Canadian uranium mining operations. However, the site specific nature of each model as well as the different assumptions made for each model does not allow a direct comparison of costs between models in spite of each facility being equally assumed as a 1,923,000 kg U (5,000,000 lbs $U_3O_8$) per year uranium mining operation.

This site-specific nature is exemplified by a comparison of net present values for Model A and Model B. The net present value calculation indicates that Model B costs are similar to Model A in spite of a much lower feed tonnage rate for the Model B processing facility. This is due to the different locations and topography assumptions for each model counteracting the benefits of treating a uranium ore of higher grade in Model B relative to the lower grade Model A ore.

Similarly, the benefits of the higher ore grade assumed for Model C is offset by the higher costs of the waste management system required for this facility as well as the higher operating costs due to its more remote location compared to Model A.

Some of the more important points highlighted from this study are as follows:

- the capital and operating costs on a dollars per tonne basis for Model A reflect the economies of scale for handling a larger quantity of ore

**1.0 (Cont'd)**

- the relatively low Model A pre-production and total capital costs are a result of lower tailings basin costs due to the fact that a natural lake basin is assumed for tailings containment

- the most significant portion of the capital costs for Model B is due to the high costs of engineered dykes for tailings containment since this model assumes a flat section of land for the tailings area

- The Model C capital costs are the highest of all three models since the waste treatment circuits include separate treatment of minewater, mill effluent and mill tailings with an ammonium sulphate crystallization plant being required for control of ammonia in effluents. Also, a layered tailings basin with an underdrainage system is used for collection of tailings water for re-treatment

- the Model C capital costs should be assessed in recognition of anticipated lower decommissioning costs for this model at the end of operations. It should be noted however that, due to the time value of money, decommissioning costs are significantly discounted in project evaluations and this should be taken into account in any future assessment

- the higher operating cost of Model C reflects the requirement of more complex waste treatment circuits for the type of ore being processed

- the remote location of northern Saskatchewan operations (Models B and C) is reflected in higher labour, reagents and power costs relative to the Ontario location (Model A)

- the use of sand filtration instead of settling ponds for clarification of treated tailings decant water can be expected to lead to higher waste management costs.

## 1.0 (Cont'd)

In summary, this cost study indicates that increasing sophistication and complexity of uranium waste management systems can be expected to lead to higher capital and operating costs. Although the time value of money must be taken into account it becomes important to consider the potential benefit of increased capital costs at the start of an operation in terms of the potential for lower decommissioning costs. This fact can also be extended to assessing total costs of waste management in terms of long-term abandonment of uranium tailings.

### FURTHER INVESTIGATIONS

As a result of this waste management cost study it has been ascertained that a more direct comparison of costs using a common site for models of uranium waste management systems would be warranted. The models used in this study were site-specific with the assumptions for each model being made to reflect typical operations in the Elliot Lake and northern Saskatchewan areas. A cost study of different waste management systems for a common site using the same design basis would provide a direct comparison for assessing the benefits of different waste treatment and waste management concepts.

It is also apparent that a more detailed financial analysis of the cost figures given in this report could be undertaken to incorporate the effect of different tax and royalty structures between provinces for uranium operations. This detailed analysis would also consider the effects of interest on borrowed money and capital depreciation.

As a follow-up to this report, studies could be initiated to evaluate other tailings disposal methods or concepts which are currently being contemplated such as deep-lake disposal and in-pit disposal. Cost estimates for these other tailings deposition methods would be required to provide a comparison to existing tailings disposal practices.

**1.0 . (Cont'd)**

The benefits of partial tailings disposal underground as backfill could also be evaluated in terms of potential cost savings as well as the potential benefits towards the long-term abandonment of uranium tailings.

The costs of monitoring uranium waste management facilities during the operating phase of a uranium facility is low in comparison to other operating costs but long-term monitoring methods and costs could possibly be investigated for abandoned and decommissioned sites.

Future studies could also include evaluations of synthetic and natural impoundment liners for uranium tailings and waste containment in terms of the relative chemical and physical properties of liners and their respective efficiencies.

The stability, acceptability and cost of different dyke construction methods could be evaluated since certain methods utilizing tailings in dyke construction may be practical from an engineering and environmental standpoint. A study to determine dyke cross-sections suitable for long-term tailings containment and seepage control could also be beneficial.

The relative clarification and sludge containment efficiency and cost of settling ponds could be compared in detail with mechanical clarification circuits in terms of costs and long-term effects on uranium waste management.

# 2.0 INTRODUCTION

## 2.1 PURPOSE OF REPORT

The National Uranium Tailings Program (NUTP) was established to provide a scientifically creditable information base on which decisions can be made, with a known degree of confidence, in establishing reasonable criteria for the protection of the environment, and which can also be used to propose practical guidelines for uranium tailings management.

As part of the disposal technology component of this program, Kilborn (Saskatchewan) Ltd., in association with Melis Consulting Engineers Ltd., were commissioned to carry out a waste management cost study under contract to the Department of Supply and Services. Assistance was provided by Knight and Piesold Ltd. with respect to the cost estimate of the tailings basin of one model in the study.

The purpose of this study was to determine the relative capital and operating costs of three different uranium waste management systems. The systems incorporate current technologies being used in Canadian uranium mining operations. This information will provide a reference point for assessment during future studies in the program.

## 2.2 SCOPE OF WORK

The scope of work for the study was to produce cost estimates to a thirty percent level of accuracy for Canadian uranium tailings management by determining all associated costs of waste management for three uranium ore processing facilities as represented by three different models. The original Statement of Work is detailed in Appendix A.

The basic requirements were to determine the total capital and operating costs of the following waste management areas for each of three Models A, B and C:

- minewater treatment
- mill effluent treatment
- tailings line
- tailings storage basin
- tailings decant treatment
- monitoring requirements.

The capital cost estimate was to include:

- equipment costs
- civil-structural costs
- construction
- engineering
- procurement
- project management
- inventory allowance
- contingencies.

**2.2 (Cont'd)**

Associated operating costs for the waste management portion of each model were to include the following:

- labour and supervision
- reagents and supplies
- services
- maintenance.

The costs excluded from this study were:

- baseline environmental studies
- environmental hearings
- general property facilities
- pumping of minewater from the mine to treatment
- uranium recovery from minewater treatment
- decommissioning.

The capital and operating costs were to be determined in 1984 dollars for a facility starting production in late 1983 with a 20-year operating life. Operating costs were to be calculated on an annual basis and expressed as dollars per tonne of ore processed, as dollars per kilogram U recovered and as dollars per pound $U_3O_8$ recovered. In all three models conservative and best practical technology and engineering was the basis of the conceptual waste treatment processes and waste management facilities.

## 2.3 BASIS OF STUDY

The original basis of study as received from the NUTP office is detailed in Appendix B. This formed the basis for detailing the three different models of Canadian uranium tailings management with minor revisions being included to reflect actual operating conditions at similar Canadian uranium mines.

In this report Model A is based on a conceptual uranium ore processing facility and waste management system typical of uranium operations in the Elliot Lake area of northern Ontario.

Model B and Model C are based on uranium operations typical of the properties located on the periphery of the Athabasca Basin in northern Saskatchewan. Model B processes a relatively clean uranium ore utilizing a conventional sub-aqueous tailings containment system. Model C is typical of a facility processing a high-arsenic uranium ore utilizing layered tailings waste management.

Each model is based on an annual uranium production of 1,923,000 kg U (5,000,000 lbs $U_3O_8$) with an operating schedule of 350 days per year and a total operating life of 20 years. The uranium recovery and mill feed grade used in the study are as follows:

Model A - 0.10% $U_3O_8$ and 95% uranium recovery
Model B - 0.475% $U_3O_8$ and 95% uranium recovery
Model C - 1.5% $U_3O_8$ and 97.5% uranium recovery

The specific design parameters for each model are provided in the detailed description of each model in subsequent sections of this report.

## 2.3 (Cont'd)

All units in this report are in the metric system and consumption figures expressed as kg/t refer to kilogram per tonne of mill feed for the ore processing facility. All costs quoted in this study are in first quarter 1984 Canadian dollars.

## 2.4 STUDY METHODOLOGY

As prime contractor for this study, Kilborn (Saskatchewan) Ltd. engaged two sub-contractors to assist in the performance of the work. Melis Consulting Engineers Ltd. was an integral part of the study team and performed the functions of study management and process engineering. Knight and Piesold Ltd. were engaged to assist in the design and costing of the Model C tailings basin. Their experience with the layered tailings concept of tailings deposition was incorporated in the design and costing of this model. Uranium and geotechnical expertise located in Kilborn's Toronto office was consulted on an advisory basis.

At the start of the program the study team met with the Scientific Authority to initiate the study. The meeting agenda included a review of the design parameters established for the three models, a review of the methods and assumptions necessary to prepare valid capital and operating cost estimates, and a review of the detailed study work plan.

Site visits to uranium operations were arranged to obtain operating information to use as a reference in the study. A site visit to Rio Algom's Elliot Lake operations was undertaken after the initial review meeting for the purpose of confirming the design basis used in Model A.

A visit to Eldor Mines' Rabbit Lake operation in northern Saskatchewan followed to confirm the Model B design basis. A visit to the Key Lake operation in northern Saskatchewan, which is a similar operation to Model C, was not made. Sufficient experience was available in the study team to provide the necessary information for this model.

**2.4 (Cont'd)**

Design work proceeded with the preparation of general process block flowsheets and detailed flowsheets for the waste treatment circuits of each model. Conceptual layouts were commenced for each model as the flowsheets were completed with the design of the mechanical and civil layouts occurring concurrently. As the conceptual layouts neared completion, quantity take-offs and capital and operating cost estimates were accomplished. With all technical activities complete, the study team concentrated on the preparation of the draft report followed by submission of the final report at the end of March 1984.

# 3.0 SUMMARY

## 3.1 RESULTS

The capital cost estimate for the waste management system of Models A, B and C are summarized in Tables 3.1, 3.3 and 3.5 respectively. These capital costs are estimated in first quarter 1984 dollars and are based on the general design parameters outlined in Section 2.3 and on the specific design presented for each model in Sections 4, 5 and 6 of this report.

The capital costs include all the construction costs associated with the waste management aspects of both minewater and mill effluent treatment for each model. This includes waste treatment circuits, reagent facilities, buildings, tailings line, tailings storage basin, tailings decant treatment, monitoring requirements, construction indirects, engineering, procurement and construction management, capital spares allowance and a contingency allowance.

A breakdown of the capital costs for each major area of waste management for Models A, B and C is presented in Tables 3.2, 3.4 and 3.6 respectively. These costs are based on the total capital costs with indirect capital costs added to the direct capital costs of each area on a proportioned basis. A sustaining capital cost is included for the Model B tailings basin in order to meet the requirement for staged construction of the tailings basin dykes as assumed in the study.

Unit capital costs relative to the annual uranium production and annual tonnage throughput for each major area are also presented in Tables 3.2, 3.4 and 3.6. These unit costs are presented in terms of dollars per annual tonne of ore processed ($/tonne), dollars per

## 3.1 (Cont'd)

annual kilogram U produced ($/kg U) and dollars per annual pound $U_3O_8$ produced ($/lb $U_3O_8$).

It should be noted that these capital cost estimates do not include any provision for pre-production construction interest since the cost of this interest is dependent on the method of project financing and the equity or credit position of the specific investor. These pre-production interest costs can have a significant impact on the total costs of any project and this should be kept in mind when reviewing the cost estimates given in this report.

The waste management-related operating costs for Models A, B and C are summarized in Tables 3.7, 3.9 and 3.11 respectively. These costs, in first quarter 1984 dollars, include all the associated costs of operating the waste management facility for each model representing a uranium processing facility producing 1,923,000 kg U (5,000,000 lbs $U_3O_8$) on an annual basis.

A breakdown of the operating costs for each major area of waste management for Models A, B and C is provided in Tables 3.8, 3.10 and 3.12 respectively. Unit operating costs relative to the annual uranium production and annual tonnage throughput for each major waste management area are also presented. These unit costs, presented in Tables 3.7 to 3.12, are in terms of dollars per tonne of ore processed ($/tonne), dollars per kilogram U produced ($/kg U) and dollars per pound $U_3O_8$ produced ($/lb $U_3O_8$).

## 3.1 (Cont'd)

A summary of the capital and operating costs as well as the total costs of the waste management system for each model for a 20-year operation is given in Table 3.13. The total cost, in constant 1984 dollars, includes pre-production capital, sustaining capital and an initial working capital allowance as well as operating costs over the 20-year life of each operation. Unit total costs are expressed in terms of dollars per tonne based on the total ore throughput of each model during the 20-year operation as well as dollars per kilogram U produced and dollars per pound $U_3O_8$ produced based on a total uranium production of 38,465,000 kg U (100,000,000 lbs $U_3O_8$) over a 20-year operating life.

A before-tax net present value (NPV), in constant 1984 dollars, is given for each model in Table 3.13. This calculation is based on the cost elements of the cash flow for each model given in Tables 4.8, 5.8 and 6.8 of this report, using a discount rate of 12.5%. It should be noted that no attempt has been made to account for the different tax and royalty structures that exist between provinces for uranium operations. Since this calculation is on a pre-tax basis, there has been no consideration given for capital cost allowance or depreciation.

## TABLE 3.1

## MODEL A
## CAPITAL COST ESTIMATE

| Direct Costs | Dollars |
|---|---|
| Minewater Treatment | 5,055,700 |
| Mill Effluent Treatment | 8,066,900 |
| Tailings Line | 714,400 |
| Tailings Storage Basin | 5,721,500 |
| Tailings Decant Treatment | 2,946,100 |
| Monitoring | 256,700 |
| Total Direct Costs | 22,761,300 |
| **Indirect Costs** | |
| Construction Indirects (8%) | 1,820,000 |
| Engineering, Procurement and Construction Management (12%) | 2,731,000 |
| Capital Spares | 562,500 |
| Subtotal | 27,875,800 |
| Contingency (10%) | 2,787,600 |
| TOTAL MODEL A ESTIMATED CAPITAL COSTS | 30,663,400 |

## TABLE 3.2

## MODEL A
## CAPITAL COSTS BY AREA

| Area | Dollars | \$/annual tonne | \$/annual kg U | \$/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 6,810,900 | 2.85 | 3.54 | 1.36 |
| Mill Effluent Treatment | 10,867,500 | 4.55 | 5.65 | 2.17 |
| Tailings Line | 962,400 | 0.40 | 0.50 | 0.19 |
| Tailings Storage Basin | 7,707,900 | 3.23 | 4.01 | 1.54 |
| Tailings Decant Treatment | 3,968,900 | 1.66 | 2.06 | 0.81 |
| Monitoring | 345,800 | 0.15 | 0.18 | 0.07 |
| TOTAL | 30,663,400 | 12.84 | 15.94 | 6.14 |

## TABLE 3.3

## MODEL B
## CAPITAL COST ESTIMATE

| Direct Costs | Dollars |
|---|---|
| Minewater Treatment | 266,000 |
| Mill Effluent Treatment | 2,511,000 |
| Tailings Line | 308,000 |
| Tailings Storage Basin | 13,301,000 |
| Tailings Decant Treatment | 2,388,000 |
| Monitoring | 282,000 |
| Total Direct Costs | 19,056,000 |
| **Indirect Costs** | |
| Construction Indirects (8%) | 1,525,000 |
| Engineering, Procurement and Construction Management (12%) | 2,287,000 |
| Capital Spares | 264,000 |
| Subtotal | 23,132,000 |
| Contingency (10%) | 2,313,000 |
| Subtotal (Pre-Production Capital Costs) | 25,445,000 |
| Sustaining Capital (Tailings Basin) | 20,200,000 |
| TOTAL MODEL B ESTIMATED CAPITAL COSTS | 45,645,000 |

## TABLE 3.4

## MODEL B
## CAPITAL COSTS BY AREA

| Area | Dollars | \$/annual tonne | \$/annual kg U | \$/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 355,200 | 0.71 | 0.19 | 0.07 |
| Mill Effluent Treatment | 3,352,900 | 6.67 | 1.74 | 0.67 |
| Tailings Line | 411,300 | 0.82 | 0.21 | 0.08 |
| Tailings Storage Basin | 17,760,500 | 35.34 | 9.23 | 3.55 |
| Tailings Decant Treatment | 3,188,600 | 6.34 | 1.66 | 0.64 |
| Monitoring | 376,500 | 0.75 | 0.20 | 0.08 |
| Subtotal (Pre-production) | 25,445,000 | 50.63 | 13.23 | 5.09 |
| Sustaining Capital (Tailings Basin) | 20,200,000 | 40.19 | 10.50 | 4.04 |
| TOTAL | 45,645,000 | 90.82 | 23.73 | 9.13 |

## TABLE 3.5

## MODEL C
## CAPITAL COST ESTIMATE

| Direct Costs | Dollars |
|---|---|
| Minewater Treatment | 3,590,000 |
| Mill Effluent Treatment | 17,041,000 |
| Tailings Line | 1,719,000 |
| Tailings Storage Basin | 11,938,000 |
| Monitoring | 1,213,000 |
| Total Direct Costs | 35,501,000 |
| **Indirect Costs** | |
| Construction Indirects (8%) | 2,840,000 |
| Engineering, Procurement and Construction Management (12%) | 4,260,000 |
| Capital Spares | 731,000 |
| Subtotal | 43,332,000 |
| Contingency (10%) | 4,333,000 |
| TOTAL MODEL C ESTIMATED CAPITAL COSTS | 47,665,000 |

## TABLE 3.6

## MODEL C
## CAPITAL COSTS BY AREA

| Area | Dollars | $/annual tonne | $/annual kg U | $/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 4,820,100 | 31.08 | 2.51 | 0.96 |
| Mill Effluent Treatment | 22,879,900 | 147.54 | 11.90 | 4.58 |
| Tailings Line | 2,308,000 | 14.88 | 1.20 | 0.46 |
| Tailings Storage Basin | 16,028,400 | 103.36 | 8.33 | 3.21 |
| Monitoring | 1,628,600 | 10.50 | 0.85 | 0.33 |
| TOTAL | 47,665,000 | 307.36 | 24.79 | 9.54 |

## TABLE 3.7

## MODEL A
## OPERATING COST ESTIMATE

| Item | Annual $ | $/tonne processed | $/kg U produced | $/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Manpower | 443,790 | 0.19 | 0.23 | 0.09 |
| Waste Treatment | 3,318,770 | 1.39 | 1.73 | 0.66 |
| Roads & Yards | 30,000 | 0.01 | 0.01 | 0.01 |
| Contingency (5%) | 189,628 | 0.08 | 0.10 | 0.04 |
| TOTAL | 3,982,188 | 1.67 | 2.07 | 0.80 |

## TABLE 3.8

## MODEL A
## ANNUAL OPERATING COSTS BY AREA

| Area | Dollars | $/tonne processed | $/kg U produced | $/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Minewater Treatment | 872,880 | 0.37 | 0.46 | 0.17 |
| Mill Effluent Treatment | 2,716,557 | 1.14 | 1.41 | 0.54 |
| Tailings Line | 26,921 | 0.01 | 0.01 | 0.01 |
| Tailings Storage Basin | 40,071 | 0.02 | 0.02 | 0.01 |
| Tailings Decant Treatment | 196,478 | 0.08 | 0.10 | 0.04 |
| Monitoring | 129,281 | 0.05 | 0.07 | 0.03 |
| TOTAL | 3,982,188 | 1.67 | 2.07 | 0.80 |

## TABLE 3.9

## MODEL B
## OPERATING COST ESTIMATE

| Item | Annual $ | $/tonne processed | $/kg U produced | $/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Manpower | 657,695 | 1.31 | 0.34 | 0.13 |
| Waste Treatment | 3,372,183 | 6.71 | 1.75 | 0.67 |
| Roads & Yards | 20,000 | 0.04 | 0.01 | - |
| Accommodation | 70,000 | 0.14 | 0.04 | 0.02 |
| Transportation | 30,000 | 0.06 | 0.02 | 0.01 |
| Contingency (5%) | 207,494 | 0.41 | 0.11 | 0.04 |
| TOTAL | 4,357,372 | 8.67 | 2.27 | 0.87 |

## TABLE 3.10

## MODEL B
## ANNUAL OPERATING COSTS BY AREA

| Area | Dollars | \$/tonne processed | \$/kg U produced | \$/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Minewater Treatment | 65,785 | 0.13 | 0.03 | 0.01 |
| Mill Effluent Treatment | 3,340,865 | 6.65 | 1.74 | 0.67 |
| Tailings Line | 63,554 | 0.13 | 0.03 | 0.01 |
| Tailings Storage Basin | 39,600 | 0.08 | 0.02 | 0.01 |
| Tailings Decant Treatment | 488,244 | 0.97 | 0.26 | 0.10 |
| Monitoring | 359,324 | 0.71 | 0.19 | 0.07 |
| TOTAL | 4,357,372 | 8.67 | 2.27 | 0.87 |

## TABLE 3.11

## MODEL C
## OPERATING COST ESTIMATE

| Item | Annual $ | $/tonne processed | $/kg U produced | $/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Manpower | 1,353,460 | 8.72 | 0.71 | 0.28 |
| Waste Treatment | 6,557,651 | 42.29 | 3.41 | 1.31 |
| Roads & Yards | 40,000 | 0.26 | 0.02 | 0.01 |
| Accommodation | 155,000 | 1.00 | 0.08 | 0.03 |
| Transportation | 65,000 | 0.42 | 0.03 | 0.01 |
| Contingency (5%) | 408,556 | 2.63 | 0.21 | 0.08 |
| TOTAL | 8,579,667 | 55.32 | 4.46 | 1.72 |

## TABLE 3.12

## MODEL C
## ANNUAL OPERATING COSTS BY AREA

| Area | Dollars | \$/tonne processed | \$/kg U produced | \$/lb $U_3O_8$ produced |
|---|---|---|---|---|
| Minewater Treatment | 1,581,064 | 10.19 | 0.82 | 0.32 |
| Mill Effluent Treatment | 6,193,698 | 39.94 | 3.22 | 1.24 |
| Tailings Line | 81,039 | 0.52 | 0.04 | 0.02 |
| Tailings Storage Basin | 162,309 | 1.05 | 0.09 | 0.03 |
| Monitoring | 561,557 | 3.62 | 0.29 | 0.11 |
| TOTAL | 8,579,667 | 55.32 | 4.46 | 1.72 |

## TABLE 3.13

## SUMMARY OF TOTAL COSTS

| | Model A | Model B | Model C |
|---|---|---|---|
| **Capital Costs** | | | |
| Pre-production ($) | 30,663,400 | 25,445,000 | 47,665,000 |
| Working Capital ($) | 300,000 | 300,000 | 500,000 |
| Sustaining Capital ($) | - | 20,200,000 | - |
| Total Capital ($) | 30,963,400 | 45,945,000 | 48,165,000 |
| **Operating Costs** | | | |
| Annual ($) | 3,982,200 | 4,357,400 | 8,579,700 |
| $/tonne processed | 1.67 | 8.67 | 55.32 |
| $/kg U produced | 2.07 | 2.27 | 4.46 |
| $/lb $U_3O_8$ produced | 0.80 | 0.87 | 1.72 |
| **Total Costs** | | | |
| Total ($) | 110,607,000 | 133,093,000 | 219,759,000 |
| $/tonne processed | 2.32 | 13.24 | 70.85 |
| $/kg U produced | 2.88 | 3.46 | 5.71 |
| $/lb $U_3O_8$ produced | 1.11 | 1.33 | 2.20 |
| Net Present Value (discount rate of 12.5%) | -44,410,000 | -46,540,000 | -81,090,000 |

## 3.2 DISCUSSION

The cost estimates detailed in the previous section reflect the waste management costs for three different uranium mine models using current waste containment methods typical of Canadian uranium mining operations. The site-specific nature of each model makes comparisons between models difficult and, in fact, it is only possible to make general observations with regards to the cost estimates given in this study.

This site-specific nature of each model is exemplified by a comparison of net present values for Model A and Model B. The net present value calculation indicates that Model B costs are similar to Model A in spite of a much lower feed tonnage rate of the Model B processing facility. The different locations and topography assumptions for each model counteract the benefits of treating a uranium ore of higher grade in Model B relative to the lower grade of Model A ore. Also, as shown in the cost estimate, it is not surprising to find that the use of a natural lake basin for tailings containment leads to much lower costs due to less requirement for engineered dykes.

Similarly, the benefits of a higher ore grade assumed for Model C is offset by the higher costs of the waste management system required for this facility as well as the higher operating costs due to its more remote location compared to Model A.

Consequently, the site specific nature of each model as well as the different assumptions made for each model does not allow a direct comparison of costs between models in spite of each facility being equally assumed as a 1,923,000 kg U (5,000,000 lbs $U_3O_8$) per year uranium mining operation.

## 3.2 (Cont'd)

Other general observations made from the results of this study include:

- the capital and operating costs on a dollars per tonne basis for Model A reflect the economies of scale for handling a larger quantity of ore
- the relatively low Model A pre-production and total capital costs are a result of lower tailings basin costs due to the fact that a natural lake basin is assumed for tailings containment
- the most significant portion of the capital costs for Model B is due to the high costs of engineered dykes for tailings containment since this model assumes a flat section of land for the tailings area
- the Model C capital costs are the highest of all three models since the waste treatment circuits include separate treatment of minewater, mill effluent and mill tailings with an ammonium sulphate crystallization plant being required for control of ammonia in effluents. Also a layered tailings basin with an underdrainage system is used for collection of tailings water for re-treatment
- the Model C capital costs should be assessed with recognition of the anticipated lower decommissioning costs for this model at the end of operations although the time value of money must be taken into account in any assessment

**3.2 (Cont'd)**

- the remote location of northern Saskatchewan operations (Models B and C) is reflected in higher labour, reagents and power costs relative to the Ontario location (Model A)

- the higher operating cost of Model C reflects the requirement of more complex waste treatment circuits for the type of ore being treated

- the use of sand filtration instead of settling ponds for clarification of treated tailings water can be expected to lead to higher waste management costs during operations.

In summary, this study shows that increasing sophistication and complexity of uranium waste management systems can be expected to lead to higher capital and operating costs. Although the time value of money must be taken into account, it becomes important to consider the potential benefit of increased capital costs at the start of an operation in terms of the potential for lower decommissioning costs. In fact the total cost of waste management should also be assessed with the inclusion of decommissioning costs in preparation for long-term abandonment.

## 3.3 FURTHER INVESTIGATIONS

As a result of this waste management cost study, it has been ascertained that a more direct comparison of costs using a common site for different methods of uranium waste management would seem to be warranted. The models used in this study were site-specific with assumptions being made to reflect typical operations in the Elliot Lake and northern Saskatchewan areas. A cost study of different waste management systems for a common site using the same design basis would provide a direct comparison for assessing the benefits of different waste treatment and waste management concepts.

It is also apparent that a more detailed financial analysis of the cost figures given in this report could be undertaken to incorporate the effect of different tax and royalty structures between provinces for uranium operations. This detailed analysis would also consider the effects of interest on borrowed money and capital depreciation.

As a follow-up to this report, studies could be initiated to evaluate other tailings disposal methods or concepts which are currently being contemplated such as deep-lake disposal and in-pit disposal. Cost estimates for these other tailings deposition methods would be required to provide a comparison to existing practices.

The benefits of partial tailings disposal underground as backfill could also be evaluated in terms of potential cost savings as well as the benefits towards the long-term abandonment of uranium tailings.

## 3.3 (Cont'd)

The costs of monitoring uranium waste management facilities during the operating phase of a uranium facility is low in comparison to other waste management costs but long-term monitoring methods and costs could possibly be investigated for abandoned and decommissioned sites.

Future studies could include evaluations of synthetic and natural impoundment liners for uranium tailings and waste containment in terms of the relative chemical and physical properties of liners and their respective efficiencies.

The stability, acceptability, and cost of different dyke construction methods could be evaluated since certain methods utilizing tailings in dyke construction may be practical from an engineering and environmental standpoint. A study to determine dyke cross-sections suitable for long-term tailings containment and seepage control could also be beneficial.

The relative clarification and sludge containment efficiency and cost of settling ponds could be compared in detail with mechanical clarification circuits in terms of costs and long-term effects on uranium waste management.

# 4.0 MODEL A STUDY

## 4.1 INTRODUCTION

The conceptual Model A uranium ore processing facility is typical of uranium operations in the Elliot Lake area of northern Ontario. The assumed mill feed grade of 0.10% $U_3O_8$ is typical for the deposits in the area with ore being mined by underground methods.

The annual uranium production of 1,923,000 kg U (5,000,000 lbs $U_3O_8$) is based on an annual mill tonnage of 2,387,350 tonnes and a 95% uranium recovery. The daily mill throughput is 7,180.0 tonnes per day based on a 350 days per year operation and 95% plant availability.

Bulk neutralization of mill tailings is practiced with tailings decant water being treated for acid neutralization, radium removal and clarification by settling ponds. Acidic minewater from underground mining operations is neutralized separately for the recovery of the contained uranium with the resulting neutralized minewater being used as process water. Tailings containment is by a conventional sub-aqueous system in a natural lake basin using open-end discharge of the tailings slurry.

The infrastructure for this model is assumed to be well established with local housing available for the employees in an established community. No company-sponsored personnel commuting is required. Line power is assumed available from Ontario Hydro at the mill site.

## 4.2 GENERAL PROCESS DESCRIPTION

The Model A ore is a quartz conglomerate typical of Elliot Lake ore with uranium mineralization consisting of brannerite and uraninite in association with pyrite.

The Model A general process flowsheet developed for this study is detailed on Drawing No. 100-10-1. It includes grinding, thickening, atmospheric leaching, partial neutralization and two-stage drum filtration. Uranium recovery is accomplished in a LAMIX (limestone-acid-magnesia ion exchange) circuit using strong-based moving-bed ion exchange with sulphuric acid elution and uranium precipitation with magnesia. A general mass balance was completed for the block flowsheet with water streams being indicated only for arriving at the final tailings density for this type of operation.

Leaching is carried out in air-agitated pachucas using sulphuric acid as the lixiviant and air as the oxidant. The leach temperature is 75°C and the leach retention time is 36 hours.

The solid tailings and waste solutions are bulk-neutralized with limestone and lime prior to discharge to the tailings containment area. Tailings decant water is treated with barium chloride at the tailings area for radium removal prior to discharge to the environment.

Minewater pumped from underground is neutralized with lime and the precipitated solids are pumped to the leaching circuit for recovery of the contained uranium. The neutralized minewater is used as process water with lake water being used for make-up as necessary.

## 4.3 MODEL A CRITERIA

The design criteria used for developing the Model A process and waste treatment flowsheets and associated general layouts were obtained from the original design basis detailed in Appendix B. Revisions were made using information obtained from a site visit to Elliot Lake uranium mining operations.

The criteria details and assumptions used for Model A are listed below.

### 4.3.1 General

| | |
|---|---|
| Annual uranium production | - 1,923,000 kg U (5,000,000 lbs $U_3O_8$) |
| Operating days per year | - 350 |
| Total years of operation | - 20 |
| Plant availability | - 95% |
| Net uranium recovery | - 95% |
| Mill feed grade (% $U_3O_8$) | - 0.10 |
| Mill throughput (t/d) | - 7,180.0 |
| Mill throughput (t/h) | - 299.2 |
| Annual mill tonnage (tonnes) | - 2,387,350 |

### 4.3.2 Process

| | |
|---|---|
| Ore grind | - 60% minus 250 microns |
| Leach sulphuric acid consumption (kg/t) | - 40 |
| Ore specific gravity | - 2.7 |
| Liquid specific gravity | - 1.0 |
| Final tailings density (% solids) | - 39 |
| Filter cake repulp density (% solids) | - 50 |

### 4.3.2 (Cont'd)

Filter discharge density (% solids) - 80
Leach discharge density (% solids) - 69
Leach feed density (% solids) - 79
Gypsum cake production (t/h) - 4
Pregnant solution flow (m$^3$/h) - 489.6
Barren diversion flow (m$^3$/h) - 22.8
IX barren wash to filtration (m$^3$/h) - 404.4
IX barren wash to gypsum cake (m$^3$/h) - 20
IX barren to tails neutralization (m$^3$/h) - 42.4
Yellowcake thickener overflow (m$^3$/h) - 42

### 4.3.3 Waste Treatment

#### 4.3.3.1 Minewater

Treatment streams (m$^3$/h):
    - minewater - 144
    - yellowcake thickener overflow - 42
    - barren diversion - 22.8
    - miscellaneous water - 20.2
    - total flow - 229
Minewater pH - 2.5 to 3.5
Precipitation retention time (h) - 1.5
Precipitation pH - 9.5
Stages - 2
Lime consumption (kg/m$^3$) - 4
Flocculant addition (g/m$^3$) - 2
Precipitate production (t/h) - 1
Clarifying thickener rise rate (m$^3$/h/m$^2$) - 3
Overflow to process water (m$^3$/h) - 215
Underflow density (% solids) - 5

### 4.3.3.2 Tailings Neutralization

| | |
|---|---|
| Treatment streams | - filter cake repulper discharge<br>- ion exchange barren<br>- excess process water<br>- sewage |
| Neutralization retention time (h) | - 6 |
| Neutralization pH | - 10.5 |
| Stages | - 3 |
| Final tailings density (% solids) | - 39 |
| Tailings flow (t/h) | - 321.9 |
| Limestone consumption (kg/t) | - 5 |
| Lime consumption (kg/t) | - 11 |

### 4.3.3.3 Tailings Decant Treatment

| | |
|---|---|
| Decant flow (m$^3$/h) | - 856 |
| Water retained in tailings (m$^3$/h) | - 144 |
| Lake water requiring treatment (m$^3$/h) | - 15 |
| Net precipitation (m$^3$/h) | - 487 |
| Decant water pH | - 4 |
| Neutralization pH | - 8.5 |
| Neutralization retention time (h) | - 0.25 |
| Radium precipitation retention time (h) | - 0.25 |
| Lime consumption (kg/m$^3$) | - 0.06 |
| Barium chloride addition (kg/m$^3$) | - 0.012 |
| Flocculant addition - 3 months/year (g/m$^3$) | - 2 |
| Settling pond retention time (h) | - 100 each |
| Number of settling ponds | - 2 |
| Precipitation (settling ponds) (m$^3$/h) | - 5 |
| Annual sludge production (dry tonnes) | - 100 |
| Net discharge to watershed (m$^3$/h) | - 861 |

### 4.3.3.4 Tailings Basin

Design life of basin - 20 years with all construction prior to mill start-up

Surge capacity - storage of 1:100 year storm or snowmelt

Decant tower and gravity drain for conveying excess basin water to decant treatment

| | |
|---|---|
| Interior and exterior dyke slopes | - 2.5:1 |
| Top width of dykes (m) | - 8 |
| Minimum freeboard (m) | - 2 |
| Lake surface area (ha) | - 175 |
| Average lake depth before placement (m) | - 15 |
| Final tailings area (ha) | - 200 |
| Final maximum tailings depth (m) | - 28 |
| Watershed area (ha) | - 700 |
| Total bulk tailings volume in pond at end of life ($m^3$) | - 50,000,000 |
| Tailings bulk density ($t/m^3$) | - 1.6 |
| Final tailings dry density ($t/m^3$) | - 1.0 |
| Specific gravity of solids | - 2.7 |
| Precipitation (mm/year) | - 960 |
| Evaporation (mm/year) | - 450 |
| Water in tailings slurry ($m^3/h$) | - 498 |
| Water retained in tailings ($m^3/h$) (based on 70% solids by weight) | - 144 |
| Runoff coefficient for lake and watershed (ratio of runoff to precipitation) | - 0.7 |

## 4.3.4 Reagents

### 4.3.4.1 Limestone

| | |
|---|---|
| Limestone to gypsum precipitation (kg/t) | - 3 |
| Limestone to partial neutralization (kg/t) | - 18 |
| Limestone to tailings neutralization (kg/t) | - 5 |
| Total limestone consumption (kg/t) | - 26 |
| Limestone grinding circuit capacity (t/h) | - 15 |
| Limestone storage (tonnes) | - 350 |
| Limestone slurry storage (days) | - 2 |
| Limestone specific gravity | - 2.6 |
| Limestone slurry density (% solids) | - 20 |
| Delivery | - bulk |

### 4.3.4.2 Lime

| | |
|---|---|
| Lime (95% CaO) consumption (kg/t): | |
| - tailings neutralization | - 11 |
| - minewater precipitation | - 3.2 |
| - decant treatment | - 0.2 |
| - total | - 14.4 |
| Lime slaking capacity (t/h) | - 7.5 |
| Lime storage (tonnes) | - 350 |
| Lime slurry storage (days) | - 4 |
| Lime specific gravity | - 2.2 |
| Lime slurry density (% solids) | - 20 |
| Delivery | - bulk, dry |

### 4.3.4.3 Barium Chloride

Barium chloride addition (kg/m$^3$) - 0.012
Barium chloride consumption (kg/t) - 0.038
Solution concentration - 5%
Delivery - bags

### 4.3.4.4 Flocculant

Flocculant addition - minewater (g/m$^3$) - 2
Flocculant addition - decant treatment (3 months/year) (g/m$^3$) - 2
Total flocculant consumption (kg/t) - 0.003
Flocculant solution concentration - 0.05%
Delivery - bags

## 4.4 WASTE TREATMENT DESCRIPTION

### 4.4.1 Minewater

The minewater treatment circuit is detailed on Drawing No. 100-10-2 with a conceptual layout shown on Drawing No. 100-10-4. Acidic minewater from underground, containing recoverable uranium values, is pumped to surface and stored in two storage tanks located in the mill building. Yellowcake thickener overflow from the uranium precipitation circuit and barren diversion from the ion exchange circuit are also pumped to the minewater storage tanks since these solutions also contain minor amounts of uranium. Miscellaneous water such as vacuum pump seal water or excess neutral thickener overflow make up the remaining streams to the minewater treatment circuit.

The water from the minewater storage tank is pumped to two precipitation tanks in series where the pH is adjusted to 9.5 with lime slurry for a total retention time of 1.5 hours. The precipitate slurry gravitates to a high-rate type thickener to effect dewatering of the precipitate sludge to 5% solids and clarification of the neutralized minewater for use as process water. The sludge is returned to the main leaching circuit to recover the contained uranium.

### 4.4.2 Tailings Neutralization

The tailings neutralization circuit is detailed on Drawing No. 100-10-2 with a conceptual layout of the equipment shown on Drawing No. 100-10-4. The filter cake discharge from the two-stage filtration-wash circuit is repulped with water to 50% solids and combined with barren solution from the ion exchange circuit in a

### 4.4.2 (Cont'd)

junction box ahead of the neutralization pachucas. The combined waste slurry is neutralized to pH 3.5 with limestone slurry followed by neutralization to pH 10.5 with lime slurry. Excess process water and domestic sewage are combined with the neutralized tailings slurry in the last neutralization pachuca and the final tailings slurry is pumped to the tailings containment area via a 2 km polyethylene tailings line. Three-stage pumping is used with standby pumps and a standby tailings line available for use as required.

Although a three-stage circuit using large pachucas is indicated for tailings neutralization in this study the retention time required and the high pulp flowrate would dictate smaller pachucas to ensure good air agitation with a total of six stages to provide the 6-hour retention time. For the purpose of this study the assumption made in the original design basis was felt adequate to generate the required cost estimate.

### 4.4.3 Decant Treatment

The decant treatment circuit is shown on Drawing 100-10-2 and the conceptual layout is detailed on Drawing No. 100-10-5. The tailings decant water for Model A is considered to be acidic (pH 4) due to the acid-generating potential of the pyritic tailings. Neutralization with lime is therefore provided ahead of radium precipitation. Lime requirements indicated in this study are based on average requirements through the life of the operation with minimal consumptions assumed for early years of operation increasing to higher levels in later years of the operation.

### 4.4.3 (Cont'd)

The decant water is neutralized to pH 8.5 in a tank with a retention time of 15 minutes. Barium chloride is then added in a second tank with the same retention time to effect radium precipitation. The treated water is clarified in two settling ponds in series with a retention time of 100 hours in each pond. Flocculant is added during periods of higher flows to improve clarification efficiency.

The treated clarified water is continuously discharged to the watershed at a rate of 861 $m^3/h$.

The ponds are sized to contain the total sludge production for a 20-year operation allowing a 25% volume loss to sludge containment through the life of the operation. An annual sludge production of 100 dry tonnes has been assumed.

The settling pond embankments are constructed of locally-available till material. The primary pond has a till-bentonite seal and the secondary pond has a synthetic liner to minimize seepage. Both ponds have an underdrainage system of sand and perforated pipes to collect and monitor seepage.

### 4.4.4 Reagents

The reagent flowsheet is described on Drawing No. 100-10-3. The conceptual layout for the limestone and lime circuit is given on Drawing No. 100-10-4. Barium chloride make-up is included in the decant treatment building (Drawing No. 100-10-5).

**4.4.4 (Cont'd)**

The limestone circuit is sized to meet limestone requirements for the total operation. Limestone is received by truck and transferred by a vibrating feeder and belt conveyor to a 350 t insulated storage bin at the mill building. The limestone circuit, with a capacity of 15 t/h, consists of a ball mill in closed-circuit with two stages of classification cyclones to provide a limestone slurry of minus 43 micron material. The limestone slurry is stored in two storage tanks and pumped to tailings neutralization and to process as a 20% solids slurry.

Lime is pneumatically unloaded to a 250 t storage bin and fed via a screw conveyor to a ball mill for slaking. The slaked lime is stored in two lime slurry tanks as a 20% solids slurry. Lime slurry is pumped as required for neutralization of waste streams in the process. Lime slurry is transported to the decant treatment area by a company-owned tank truck and unloaded to a small storage tank at the decant treatment building.

Barium chloride is received in bags and stored at the decant treatment building. It is manually made up as a 5% solution in a small mix tank and transferred to a holding tank from which it is added to the precipitation tank via metering pumps.

Minewater flocculant is made up in the mill using an aspirator and process water to provide a stock solution of 0.5% concentration. It is further diluted in a day tank to a 0.05% solution prior to use in the minewater clarifying thickener.

The decant treatment settling pond flocculant is made up as required in a small mix tank at the decant treatment building.

**4.4.4 (Cont'd)**

Air supply is provided by two 2,000 $m^3/h$ air compressors for agitation in the tailings pachucas, minewater precipitation tanks, lime slurry tanks and for miscellaneous agitation. Standby air and instrument air have been assumed to be available from the main process plant compressors.

Process water requirements are minimal for the waste treatment circuits and the water supply for reagent make-up is assumed available from the main process plant water supply.

## 4.5 TAILINGS CONTAINMENT

Tailings storage for Model A is within a typical Elliot Lake area lake basin. Approximately one-half of the total tailings volume over the design life is stored below the original water level with the remainder above and around the lake surface. In this model, it is assumed that the lake exists within a steep-walled basin and inlet and outlet channels requiring dykes are through narrow, steep-sided valleys. These assumed conditions result in minimal dyke construction. The site plan and details of the Model A storage basin are shown on Drawing Nos. 100-35-1 and 100-35-2.

Three horizontal centrifugal pumps, in series, discharge tailings through a polyethylene tailings pipeline to the basin. Standby pumps and duplicate tailings lines are included to ensure uninterrupted service for scheduled repairs or in case of emergency shutdown. The pipes are supported on a bench adjacent to the tailings access road. Wooden posts and cross-members are utilized to provide tie-down supports for the pipelines. Discharge velocity and flow is monitored in the tailings lines at the mill and basin in order to detect leaks in the pipeline. Open-end tailings discharge into the upper reaches of the basin is practiced. The typical angle of repose for tailings deposited in this manner is between 0.5 and 1.0 percent.

It is assumed that all construction and required structures are completed prior to mill start-up. This includes:

- three small dykes
    - 100 m long and 15.5 m high
    - 50 m long and 9.5 m high
    - 20 m long and 4.5 m high

**4.5 (Cont'd)**

- grouting of bedrock below dyke bases. There is no provision for grouting bedrock in other areas of the basin since all foundation rock is assumed to be sufficiently impermeable to keep seepage within required guidelines
- site clearing for borrow pits, dykes and areas of tailings deposition above existing lake level
- six kilometres of permanent access roads
- three kilometres of temporary construction roads
- one kilometre freshwater diversion ditching
- emergency spill pond for the tailings lines at the mill site. The spill pond has capacity to store four hours of average tailings flow
- five kilometre power supply to decant treatment, 3 phase, overhead, including substation.

The construction of the three small dyke sections includes a coarse till shell, low permeability till core, sand filter drain and interior and exterior slope protection. It is assumed that both coarse till and fine-grained impermeable till are available within a 3 km haul distance of the proposed basin. If a suitable low permeability till material was not available for core construction then a synthetic liner installed either in vertical accordion fashion or along the interior dyke slope could be utilized to control seepage. For the purposes of this study it is assumed that

## 4.5 (Cont'd)

low-permeability till is readily available and that costs associated with either method would not differ significantly from those included for till core construction.

Sampling piezometers are installed in the dykes and foundation rock to monitor and sample seepage. The dyke filter drains include a perforated pipe collection network and sump for additional seepage monitoring. Piezometers are also installed in locations surrounding the basin to detect and monitor seepage.

Excess water is released from the tailings basin through a decant tower and gravity pipeline to the decant treatment building. This building is located at the downstream end of the lake since tailings discharge is at the upper end of the lake basin. The average decant flow includes net precipitation, water in the tailings slurry and lake water, less the water retained in the tailings solids.

Basin design allows for the storage of extreme storm or snowmelt events within the basin with subsequent metering to decant treatment. In order to reduce the total volume of water requiring treatment approximately 85 percent of the initial lake volume would be pumped to the downstream watercourse prior to initial tailings deposition.

## 4.6 MONITORING REQUIREMENTS

The environmental monitoring requirements for Model A have been assumed to be typical of the requirements detailed by the Ontario Ministry of Environment (MOE) and the Atomic Energy Control Board of Canada (AECB).

The sample locations, the constituent analyses and the frequency of sampling are detailed in Table C-1 of Appendix C. The sample locations assumed in Model A include:

- decant water (1)
- treated effluent (1)
- minewater (1)
- settling pond underdrains (1)
- local creek (1)
- downstream stations (3).

The constituent analyses, summarized in Table C-2, are typical of those prescribed by MOE. A total of approximately 5,000 analyses and measurements are required on an annual basis. It is assumed that all analyses are carried out on site in the assay office servicing the total operation.

## 4.7 CAPITAL COSTS

### 4.7.1 Summary

The cost summary presented in Table 4.1 provides the estimated capital costs of the Model A waste management system for a 7,180 t/d ore processing facility. These estimated costs total $30,663,400 in first quarter 1984 Canadian dollars and provide for all costs of the waste management aspects of both minewater and mill tailings treatment.

The estimate includes:

- building structures
- minewater treatment circuit
- mill tailings treatment circuit including reagent make-up facilities
- proportioned costs of a limestone circuit
- tailings lines
- tailings storage basin
- tailings decant treatment
- monitoring
- indirect costs of construction
- engineering, procurement, and construction management
- capital spares inventory
- contingency allowance.

The capital cost estimate details for Model A are provided under separate cover.

## TABLE 4.1

## MODEL A
## CAPITAL COST ESTIMATE

| Direct Costs | Dollars |
|---|---|
| Minewater Treatment | 5,055,700 |
| Mill Effluent Treatment | 8,066,900 |
| Tailings Line | 714,400 |
| Tailings Storage Basin | 5,721,500 |
| Tailings Decant Treatment | 2,946,100 |
| Monitoring | 256,700 |
| Total Direct Costs | 22,761,300 |
| **Indirect Costs** | |
| Construction Indirects (8%) | 1,820,000 |
| Engineering, Procurement and Construction Management (12%) | 2,731,000 |
| Capital Spares | 562,500 |
| Subtotal | 27,875,800 |
| Contingency (10%) | 2,787,600 |
| TOTAL MODEL A CAPITAL COSTS | 30,663,400 |

### 4.7.1 (Cont'd)

A breakdown of the capital costs for each major area is presented in Table 4.2. These costs are based on the total capital costs with indirect capital costs added to the direct capital costs of each area on a proportioned basis. Unit capital costs relative to the annual uranium production and annual tonnage throughput of the facility are also presented by area.

On this basis, the total unit capital costs for Model A are:

- $12.84 per annual tonne of ore processed
- $15.94 per annual kilogram of U produced
- $6.14 per annual pound of $U_3O_8$ produced.

These cost estimates do not include any provision for pre-production construction interest as the total value of this interest is dependent on the method of financing and the equity or credit position of the specific investor. These pre-production interest costs can have a significant impact on the total costs of any project and this should be kept in mind when reviewing the cost estimates given in this report.

### 4.7.2 Basis of Estimate

The capital cost estimate is based on the following information:

- flowsheets and conceptual layouts prepared by Kilborn and Melis Consulting Engineers Ltd.
- process and material specifications as supplied by Melis Consulting Engineers Ltd.
- estimated construction material quantities as derived by Kilborn

## TABLE 4.2

## MODEL A
## CAPITAL COSTS BY AREA

| Area | Dollars | \$/annual tonne | \$/annual kg U | \$/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 6,810,900 | 2.85 | 3.54 | 1.36 |
| Mill Effluent Treatment | 10,867,500 | 4.55 | 5.65 | 2.17 |
| Tailings Line | 962,400 | 0.40 | 0.50 | 0.19 |
| Tailings Storage Basin | 7,707,900 | 3.23 | 4.01 | 1.54 |
| Tailings Decant Treatment | 3,968,900 | 1.66 | 2.06 | 0.81 |
| Monitoring | 345,800 | 0.15 | 0.18 | 0.07 |
| TOTAL | 30,663,400 | 12.84 | 15.94 | 6.14 |

4.7.2 (Cont'd)

- budget quotations on major equipment
- pricing data from Kilborn's files and records.

For purposes of this study, it has been assumed that:

- all construction services would be tendered by fixed price contract. The average labour rate is $35.00 per hour based on a 40-hour work week. This labour rate includes provision for direct wages, payroll burdens and benefits, small tools and consumables, site supervision, and the contractor's overhead and fee

- construction labour force is available in the immediate surrounding area

- all costs are based on first quarter 1984 Canadian dollars

- the cumulative price factor applied to base prices for process equipment is eighteen percent. This includes:

  - provincial sales tax - 7%
  - freight allowance - 5%
  - miscellaneous minor materials allowance - 5%

No allowances have been made in the capital estimate for:

- escalation beyond the first quarter 1984
- pre-production construction interest
- working capital
- production equipment or materials

4.7.2 (Cont'd)

- purchase of land or right-of-way easements
- start-up assistance
- cost of pre-appropriation studies, development work or environmental studies and hearings
- owners' costs such as an owners' representative
- general property facilities such as housing and administration offices
- pumping minewater from the mine to the treatment facility
- decommissioning.

### 4.7.3 Methodology

Equipment prices were obtained on an individual basis from budget quotations and historical data. Mechanical installation costs, including field material, were factored from the equipment costs. In-plant piping, electrical and instrumentation costs were factored from the total installed mechanical costs. Building structure costs for each area were based on volumes and volume unit prices derived from historical data.

The limestone circuit is sized and priced for the total mill requirements including both production and waste management. Twenty percent of this total cost was proportioned to the waste management area based on the percentage limestone usage for waste management.

The monitoring cost includes piezometers and sampling wells for the tailings area as well as the portion of the capital cost of the laboratory building and equipment dedicated to waste management. The assumption was made that thirty percent of analytical efforts for a uranium operation are dedicated to environmental requirements.

### 4.7.3 (Cont'd)

The allowance for construction indirects has been estimated as eight percent of the total direct costs. It includes:

- temporary site services
- permits, insurance and bonds
- winter works
- rental of mobile equipment
- on-site construction supervision facilities
- pre-construction geotechnical and hydrogeotechnical investigations.

The engineering, procurement and construction management function has been estimated as twelve percent of the direct costs. As well as design engineering, this includes provision for project management, cost control, scheduling, contract administration and inspection services.

An allowance of five percent of home office costs has been included for capital spares inventory. The contingency allowance provides for additions and unidentified items within the stated scope of work but it is not intended to cover major changes in scope.

## 4.8 OPERATING COSTS

### 4.8.1 Summary

The waste management related operating costs for Model A, based on the design basis detailed in Section 4.3, total \$3,982,188 annually in first quarter 1984 Canadian dollars, including a five percent contingency factor. This annual operating cost is equivalent to \$1.67/tonne of ore processed or \$2.07/kg U (\$0.80/lb $U_3O_8$) based on 1,923,000 kg U (5,000,000 lbs $U_3O_8$) of recovered uranium per year.

A summary of the operating costs is presented in Table 4.3 and an estimated breakdown of the costs for each area of the waste management system is provided in Table 4.4. Cost details are given in Tables 4.5, 4.6 and 4.7.

The operating cost breakdown given in Table 4.4 is a proportioned estimate of the total costs allocated to the following areas:

- minewater treatment
- mill effluent treatment
- tailings line
- tailings basin
- tailings decant treatment
- monitoring.

Cost of reagents and supplies have been proportioned according to the estimated usage for each area.

The minewater treatment operating costs include all costs associated with the treatment of minewater. The mill effluent treatment costs

### 4.8.1 (Cont'd)

include the costs of treating all mill waste solutions. The decant treatment costs include the cost of treating tailings decant water at the tailings area. Monitoring costs are the costs associated with environmental sampling and analysis. The tailings line and tailings basin costs are the operating costs associated with these two areas.

### 4.8.2 Manpower

The annual manpower costs of $443,790 detailed in Table 4.5 includes the manpower specifically required by the waste management section of the uranium milling operation. The indicated hourly rates are typical of union rates for the area with a twenty-five percent payroll burden assumed for the estimate.

The supervision allowance for operations is based on a supervisor spending twenty percent of his time, three shifts per day, on operating areas related to waste management. The remaining supervision allowance for maintenance, analytical, environmental, and a process engineer is based on dayshift supervision only.

To meet the sampling requirements for environmental monitoring an environmental sampler is assumed to spend thirty percent of his time collecting environmental samples as listed in Appendix C. He is assumed to be working on assignments unrelated to waste management (air sampling etc.) the remainder of the time. A total of 1.5 analytical technicians per day is assumed to be sufficient manpower for meeting the requirements for environment-related analysis.

## TABLE 4.3

## MODEL A
## OPERATING COST SUMMARY

| Item | Annual $ | $/tonne | $/kg U | $/lb $U_3O_8$ |
|---|---|---|---|---|
| Manpower | 443,790 | 0.19 | 0.23 | 0.09 |
| Waste Treatment | 3,318,770 | 1.39 | 1.73 | 0.66 |
| Roads & Yards | 30,000 | 0.01 | 0.01 | 0.01 |
| Contingency (5%) | 189,628 | 0.08 | 0.10 | 0.04 |
| TOTAL | 3,982,188 | 1.67 | 2.07 | 0.80 |

## TABLE 4.4

## MODEL A
## OPERATING COSTS BY AREA ($)

| Item | Minewater Treatment | Mill Effluent Treatment | Tailings Line | Tailings Basin | Tailings Decant Treatment | Monitoring | Total |
|---|---|---|---|---|---|---|---|
| Manpower | 109,830 | 164,747 | 12,659 | 12,523 | 34,906 | 109,125 | 443,790 |
| Reagents & Supplies | 606,469 | 2,092,496 | 7,980 | 5,640 | 112,992 | 10,000 | 2,835,577 |
| Power | 56,503 | 260,214 | - | - | 31,235 | 3,000 | 350,952 |
| Heating | 58,512 | 69,740 | - | - | 2,989 | 1,000 | 132,241 |
| Roads & Yards | - | - | 5,000 | 20,000 | 5,000 | - | 30,000 |
| SUB-TOTAL | 831,314 | 2,587,197 | 25,639 | 38,163 | 187,122 | 123,125 | 3,792,560 |
| Contingency (5%) | 41,566 | 129,360 | 1,282 | 1,908 | 9,356 | 6,156 | 189,628 |
| TOTAL | 872,880 | 2,716,557 | 26,921 | 40,071 | 196,478 | 129,281 | 3,982,188 |
| $/tonne | 0.37 | 1.14 | 0.01 | 0.02 | 0.08 | 0.05 | 1.67 |
| $/kg U | 0.46 | 1.41 | 0.01 | 0.02 | 0.10 | 0.07 | 2.07 |
| $/lb $U_3O_8$ | 0.17 | 0.54 | 0.01 | 0.01 | 0.04 | 0.03 | 0.80 |

## TABLE 4.5

## MODEL A
## ANNUAL MANPOWER COSTS

| Employee | Requirement | Number | Rate | Annual $ |
|---|---|---|---|---|
| Supervision: | | | | |
| - Operational | 0.2 man/shift | 0.8 | $3,500/month | 33,600 |
| - Maintenance | 0.2 man/day | 0.2 | $3,500/month | 8,400 |
| - Environmental | 0.5 man/day | 0.5 | $3,500/month | 21,000 |
| - Analytical | 0.2 man/day | 0.2 | $3,250/month | 7,800 |
| - Process Engineer | 0.2 man/day | 0.2 | $3,250/month | 7,800 |
| Environmental Sampler | 0.3 man/day | 0.3 | $2,500/month | 9,000 |
| Analytical Technician | 1.5 men/day | 1.5 | $2,750/month | 49,500 |
| Waste Treatment Operator | 0.5 man/shift | 2.0 | $15.00/hour | 63,000 |
| Reagent Operator | 0.5 man/shift | 2.0 | $14.00/hour | 58,800 |
| Tails Area Operator | 0.8 man/day | 0.8 | $14.00/hour | 22,400 |
| Waste Treatment Area Maintenance | 40 mandays/month | 2.0 | $16.00/hour | 61,440 |
| Tails Area Maintenance | 8 mandays/month | 0.4 | $16.00/hour | 12,290 |
| SUB-TOTAL | | 10.9 | | 355,030 |
| Payroll Burden (25%) | | | | 88,760 |
| TOTAL | | | | 443,790 |

## TABLE 4.6

## MODEL A
## WASTE TREATMENT COSTS

| Item | Annual $ | $/tonne | $/kg U | $/lb $U_3O_8$ |
|---|---|---|---|---|
| Reagents & Supplies | 2,835,577 | 1.19 | 1.48 | 0.56 |
| Power | 350,952 | 0.15 | 0.18 | 0.07 |
| Heating | 132,241 | 0.05 | 0.07 | 0.03 |
| TOTAL | 3,318,770 | 1.39 | 1.73 | 0.66 |

## TABLE 4.7

## MODEL A
## REAGENTS AND SUPPLIES

| Item | Unit Cost ($/t) | Consumption | | Annual $ |
|---|---|---|---|---|
| | | kg/t* | t/a | |
| Limestone | 15.50 | 5.0 | 11,937 | 185,024 |
| Lime (95% CaO) | 69.20 | 14.400 | 34,378 | 2,378,958 |
| Barium Chloride | 415 | 0.038 | 90 | 37,350 |
| Flocculant | 4,500 | 0.003 | 7.61 | 34,245 |
| Laboratory Supplies | | | | 10,000 |
| Maintenance Supplies | | | | 150,000 |
| Operating Supplies | | | | 30,000 |
| Lime Slurry Trucking | | | | 10,000 |
| TOTAL | | | | 2,835,577 |

* Reagent consumption based on 2,387,350 tonnes mill feed per year.

## 4.8.2 (Cont'd)

Operating manpower for waste treatment and reagent circuits are assumed at one man per shift, 350 days per year, 24 hours per day. A dayshift tailings area operator is assumed to spend eighty percent of his time checking the tailings area and monitoring the decant treatment circuit as well as making up reagents as required. A maintenance allowance has been included with 48 mandays per month being assumed for maintenance in the waste treatment and tailings area.

## 4.8.3 Reagents and Supplies

The cost of reagents is based on the consumption rates given in the design basis and the delivered price of reagents to the Model A processing facility. The annual consumption is based on 2,387,350 tonnes of mill feed per year.

Limestone consumption includes the requirement for tailings neutralization and excludes the limestone used for uranium recovery. The total lime costs include the requirements for tailings neutralization, minewater precipitation, and decant treatment. Barium chloride costs are for decant treatment only while flocculant costs include the requirements for seasonal use at the decant treatment area and the annual requirements for minewater precipitation. The cost of lime haulage to the decant treatment building is a lump sum estimate for the anticipated annual lime requirement for that area.

Lump sum estimates have been used for laboratory, maintenance, and and operating supplies sufficient to meet the requirements for operating the waste management section of the uranium processing operation.

### 4.8.4 Power

The total power demand for operating the waste treatment circuits, reagent facilities, decant treatment circuit and the tailings area is estimated at 2,089 kW (2,800 HP). The total annual power cost is estimated at $350,962 assuming an 85% power efficiency and a power cost of $0.017/kWh for a 350 day per year operation.

### 4.8.5 Heating

The annual heating costs of $132,241 for the waste treatment building, a portion of the laboratory building and the decant treatment building are based on 629,720 L of propane at $0.21/L.

### 4.8.6 Roads and Yards

A lump sum estimate of $30,000 per year is included for road and yard maintenance of the tailings area and the general waste treatment site.

## 4.9 CASH FLOW AND NET PRESENT VALUE

A net cash flow for the Model A waste management system is presented in Table 4.8. This cash flow is based on the construction schedule indicated in the terms of reference for this study. The key dates are:

- Engineering Start: January 1980
- Construction Start: June 1981
- Plant Startup: August 1983
- Plant decommissioning: 2004

It has also been assumed that no significant production would occur during the period August 1983 to December 1983 and there would therefore be no significant operating costs during this period.

Normally a total cash flow is the summation of revenue, operating costs, interest expenses, freight and transportation, fixed capital, working capital, sustaining capital, federal taxation, provincial taxation and selling expenses.

Since this study presents a net cash flow for waste management only, the net cash flow is defined as the total project cash flow including the waste management costs minus the total project cash flow excluding the waste management costs. Therefore, in the net cash flow calculation, all costs that are not affected by waste management, such as revenue, freight, transportation and selling expenses, can be excluded from consideration.

**4.9 (Cont'd)**

Since the study criteria also excluded taxation considerations federal and provincial taxes have been excluded in the calculations. In addition, since interest expenses are dependent on financing and are investor-specific they are also excluded in this analysis.

Consequently, the net cash flow equation used for calculating the incremental waste management costs is:

Cash Flow = operating costs + fixed capital + working capital + sustaining capital.

For Model A sustaining capital is negligible and it does not enter into the calculation.

A net present value of minus $44,410,000 has been determined for Model A based on the cost elements of the net cash flow given in Table 4.8. This net present value is based on a discount rate of 12.5% and gives the net present value in 1984 constant dollars at the date in time when the initial capital is committed (i.e. 1980). Since this calculation is on a pre-tax basis, no consideration is given for capital cost allowance or depreciation.

# TABLE 4.8

## MODEL A

## CASH FLOW

(CONSTANT 1984 $ x 1,000,000)

| | 1980 | 1981 | 1982 | 1983 | 1984 | 1985 | 1986 | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | 2000 | 2001 | 2002 | 2003 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 |
| Operating Cost | | | | | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 |
| Capital Cost: | | | | | | | | | | | | | | | | | | | | | | | | |
| - initial | 0.7 | 6.0 | 12.0 | 12.0 | | | | | | | | | | | | | | | | | | | | |
| - sustaining | | | | | | | | | | | | | | | | | | | | | | | | |
| Working Capital | | | | 0.3 | | | | | | | | | | | | | | | | | | | | |
| Total Costs | 0.7 | 6.0 | 12.0 | 12.3 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 | 4.0 |
| Present Value Factor | 1.0 | 0.89 | 0.79 | 0.70 | 0.62 | 0.55 | 0.49 | 0.44 | 0.39 | 0.35 | 0.31 | 0.27 | 0.24 | 0.22 | 0.19 | 0.17 | 0.15 | 0.14 | 0.12 | 0.11 | 0.09 | 0.08 | 0.07 | 0.07 |
| Present Value | 0.7 | 5.34 | 9.48 | 8.61 | 2.48 | 2.20 | 1.96 | 1.76 | 1.56 | 1.40 | 1.24 | 1.08 | 0.96 | 0.88 | 0.76 | 0.68 | 0.60 | 0.56 | 0.48 | 0.44 | 0.36 | 0.32 | 0.28 | 0.28 |
| Net Present Value | 0.7 | 6.04 | 15.52 | 24.13 | 26.61 | 28.81 | 30.77 | 32.53 | 34.09 | 35.49 | 36.73 | 37.81 | 38.77 | 39.65 | 40.41 | 41.09 | 41.69 | 42.25 | 42.73 | 43.17 | 43.53 | 43.85 | 44.13 | 44.41 |

# 5.0 MODEL B STUDY

## 5.1 INTRODUCTION

The location of the Model B uranium ore processing operation for this study is the Athabasca Basin area of northern Saskatchewan. The ore being processed is assumed to be low in arsenic and heavy metal concentration with an average uranium grade of 0.475% $U_3O_8$. The majority of the ore is from underground mining operations with only a quarter of the mill feed coming from an open-pit operation.

The annual uranium production of 1,923,000 kg U (5,000,000 lbs $U_3O_8$) is based on an annual mill tonnage of 502,607 tonnes and a 95% uranium recovery. The daily mill throughput is 1,511.6 tonnes per day based on a 350 days per year operation and 95% plant availability.

Bulk neutralization of the mill tailings is practiced with the tailings decant water being treated for radium removal and clarification by sand filtration. Minewater is clarified and collected in a clarification pond prior to use as process water. Tailings containment is by a sub-aqueous method in a tailings basin constructed of peripheral engineered dykes.

It has been assumed that there is an all-weather road to the property but the remote location necessitates the use of diesel-generated power on site. On-site accommodation is provided for the employees as well as company-sponsored air transportation for operating personnel to commute from established communities on a 7-day in, 7-day out work schedule.

## 5.2 GENERAL PROCESS DESCRIPTION

The mineralogy of the ore being processed in Model B consists of pitchblende and uranophane uranium mineralization with quartzite and dolomite.

The Model B general process flowsheet is detailed on Drawing No. 100-10-6. The milling process includes grinding, cyclone classification, atmospheric leaching, liquid/solid separation with counter-current decantation (CCD) thickeners, clarification, solvent extraction, uranium precipitation and bulk tailings neutralization.

Leaching is carried out in mechanically-agitated tanks using sulphuric acid and sodium chlorate at a temperature of 60°C and a leach retention time of 10 hours. Recovery of pregnant solution and washing of the leach residue is accomplished in six stages of CCD thickeners. Clarification of the pregnant solution ahead of solvent extraction is accomplished by sand filters.

The loaded organic from solvent extraction is stripped using a strong sulphuric acid solution to produce a high-concentration pregnant strip liquor. Raffinate is recycled for use as wash in the CCD circuit. A bleed stream of the stripped organic is regenerated with sodium carbonate prior to re-use in the extraction circuit. Spent sodium carbonate solution is pumped to the CCD circuit for recovering minor amounts of uranium.

Pregnant strip liquor is treated with lime, ferric sulphate and magnesia to precipitate contaminants ahead of uranium precipitation. The resulting iron cake precipitate is repulped and leached with sulphuric acid prior to being recycled to the liquid/solid separation circuit.

**5.2 (Cont'd)**

Uranium precipitation is carried out with hydrogen peroxide using magnesia for pH control. Barren strip liquor from the uranium precipitation circuit is clarified in sand filters prior to disposal to the tailings neutralization circuit. Yellowcake is washed in a thickener-centrifuge circuit then dried and packaged for shipment.

The CCD tails report to the bulk neutralization circuit along with a bleed raffinate stream, the barren strip solution from uranium precipitation, excess process water and domestic sewage from the site operations. The total tailings are neutralized with lime prior to disposal to the tailings area. Tailings pond decant water is treated in a separate facility at the tailings area.

Minewater pumped from the underground and open-pit operations is clarified in a pond and used as process water in the mill circuit. Any excess water is directed to the tailings neutralization circuit.

## 5.3 MODEL B CRITERIA

The design criteria used for Model B was based on the original design basis detailed in Appendix B with minor revisions made to reflect the conditions of a typical operation in northern Saskatchewan. A description of the design basis used in the study follows.

### 5.3.1 General

Annual uranium production - 1,923,000 kg U (5,000,000 lbs $U_3O_8$)
Total years of operation - 20
Plant availability - 95%
Operating days per year - 350
Net uranium recovery - 95%
Mill feed grade (% $U_3O_8$) - 0.475
Mill throughput (t/d) - 1,511.6
Mill throughput (t/h) - 63.0
Annual mill tonnage (tonnes) - 502,607

### 5.3.2 Process

Ore grind - 50% passing 74 microns
Sodium chlorate addition (kg/t) - 8.25
Leach sulphuric acid consumption (kg/t) - 60
Weight loss in leach - 5%
Ore specific gravity - 2.7
Leach liquor specific gravity - 1.03
Pregnant solution specific gravity - 1.01
Pregnant strip liquor specific gravity - 1.30

## 5.3.2 (Cont'd)

Leach discharge density (% solids) - 50
CCD tails density (% solids) - 45
Pregnant solution/ore ratio - 2.64/1
Pregnant strip liquor flow ($m^3/h$) - 2.2
Lime to iron cake precipitation (kg/t) - 15
Iron cake production (t/h) - 4
Iron cake re-dissolution (t/h) - 1
Organic regeneration - 20% of flow
Raffinate recycle to CCD's - 96%
Barren strip flow ($m^3/h$) - 23

## 5.3.3 Waste Treatment

### 5.3.3.1 Minewater

Flowrate range ($m^3/h$) - 54 to 108
Average flow ($m^3/h$) - 81
Suspended solids (mg/L) - 260
Clarification Pond Settling Time (d) - 7
Annual slimes production (tonnes) - 177
Minewater pH - 6
Minewater flow to process ($m^3/h$) - 45.4
Excess minewater to tailings ($m^3/h$) - 35.6

### 5.3.3.2 Tailings Neutralization

Treatment streams - CCD tailings
- Raffinate
- Barren strip liquor
- Excess minewater
- Sewage

### 5.3.3.2 (Cont'd)

Neutralization retention time (h) - 6
Neutralization pH - 8.5
Final tailings density (% solids) - 29
Tailings flow (t/h) - 65.7
Lime consumption (kg/t) - 30

### 5.3.3.3 Tailings Decant Treatment

Water retained in tailings ($m^3/h$) - 29
Net precipitation ($m^3/h$) - 15.3
Decant flow ($m^3/h$) - 151.2
Barium chloride addition ($kg/m^3$) - 0.1
Flocculant addition ($g/m^3$) - 2
Total retention time (h) - 1
Sand filtration unit flowrate ($m^3/h/m^2$) - 15
Number of filters - 3 operating, plus 1 standby or backwash
Backwash flow - 4% of feed volume
Nitric acid wash - semi-annual
Net discharge to watershed ($m^3/h$) - 146.2
Discharge pH - 7

### 5.3.3.4 Tailings Basin

Design life - 20 years with staged construction of basin dykes
Initial dyke construction of 3.5 meters with bi-annual dyke raising of approximately 1.3 meters

### 5.3.3.4 (Cont'd)

| | |
|---|---|
| Storage volume adequate to store maximum snowmelt or 1:100 year storm | |
| Final dyke height (m) | - 15 |
| Interior slopes | - 3:1 |
| Exterior slopes | - 2.8:1 |
| Top width of dykes (m) | - 8 |
| Minimum freeboard (m) | - 2 |
| Basin area (ha) | - 88 |
| Watershed area (ha) | - 88 |
| Maximum tailings depth (m) | - 13 |
| Total bulk tailings volume in pond at end of life ($m^3$) | - 11,500,000 |
| Tailings bulk density ($t/m^3$) | - 1.6 |
| Final tailings dry density ($t/m^3$) | - 0.96 |
| Specific gravity of solids | - 2.7 |
| Precipitation (mm/a) | - 413 |
| Evaporation (mm/a) | - 375 |
| Water in tailings slurry ($m^3/h$) | - 159.8 |
| Water retained in tailings ($m^3/h$) (based on 60% solids by weight) | - 29 |
| Runoff coefficient for basin | - 1.0 |

### 5.3.4 Reagents

#### 5.3.4.1 Lime

| | |
|---|---|
| Lime (95% CaO) consumption (kg/t): | |
| - tailings neutralization | - 30 |
| - iron cake precipitation | - 15 |
| - total | - 45 |

### 5.3.4.1 (Cont'd)

Lime slaking capacity (t/h) - 5
Lime storage (tonnes) - 600
Lime slurry storage (days) - 2
Lime specific gravity - 2.2
Lime slurry density (% solids) - 20
Delivery - bulk, dry

### 5.3.4.2 Barium Chloride

Barium chloride addition (kg/$m^3$) - 0.1
Barium chloride consumption (kg/t) - 0.26
Solution concentration - 5%
Delivery - bags

### 5.3.4.3 Flocculant

Flocculant addition (g/$m^3$) - 2
Flocculant consumption (kg/t) - 0.005
Solution concentration - 0.05%
Delivery - bags

### 5.3.4.4 Nitric Acid

Nitric acid supply concentration - 65 to 70%
Nitric acid solution concentration - 4%
Nitric acid consumption (kg/t) - 0.1
Sand filter acid wash frequency - semi-annual
Number of bed volumes per acid wash - 4
Delivery - bulk

## 5.4 WASTE TREATMENT DESCRIPTION

### 5.4.1 Minewater

The only treatment required for Model B minewater is removal of suspended solids in a clarification pond as shown on Drawing No. 100-10-7.

Minewater from underground and open-pit operations, containing on average 260 mg/L suspended solids, is pumped to a synthetic-lined clarification pond with 7 days retention time and reclaimed through a decant pipe for pumping to the process water tank in the mill. The solids collect at the bottom of the pond and are removed once a year by pumping to the grinding circuit in the mill. At an average flowrate of 81 $m^3/h$ and 260 mg/L suspended solids concentration a total of 177 tonnes (dry basis) of solids would be collected on an annual basis. Excess minewater reporting to the process water tank would be directed to the tailings neutralization circuit for treatment and disposal to the tailings area.

### 5.4.2 Tailings Neutralization

Bulk neutralization of waste streams is practiced in Model B as shown on Drawing No. 100-10-7. The conceptual equipment layout is shown on Drawing No. 100-10-9.

The washed leach residue from the CCD circuit (CCD tails) is combined in a junction box with a raffinate bleed stream from the solvent extraction circuit and the barren strip solution from the yellowcake precipitation circuit. The combined waste slurry is neutralized with lime to pH 8.5 in a series of three mechanically-agitated tanks providing a total of 6 hours retention

### 5.4.2 (Cont'd)

time. Excess process water and domestic site sewage report to the last neutralization tank to make up the total final tailings for discharge to the tailings area. The neutralized tailings slurry at a rate of 65.7 t/h and at 29% solids is pumped to the tailings area using a two-stage pumping circuit and a 2 km polyethylene tailings line. Both pumps and line are duplicated to ensure continuous service.

### 5.4.3 Decant Treatment

The decant treatment circuit for Model B, shown on Drawing No. 100-10-7, includes treatment for radium removal and clarification of the final effluent. A conceptual layout of the decant treatment building is shown on Drawing No. 100-10-10.

Tailings decant water is reclaimed from the tailings area using vertical pumps mounted on a barge. Decant water, at a rate of 151.2 $m^3/h$, is pumped to the first treatment tank in the decant treatment building where barium chloride solution is added to effectively precipitate the radium in the water. Flocculant is added in a second tank to effect coagulation of the suspended solids prior to clarification in a sand filtration circuit.

Dual-media sand filters are used to provide a high-clarity final effluent for discharge to the environment. The sand filters are backwashed at regular intervals with clarified effluent and the backwash water, containing trace amounts of precipitates, is returned to the tailings basin. The final effluent is continually discharged to the local watershed at a rate of 146.2 $m^3/h$.

### 5.4.3 (Cont'd)

A system has been included for acid washing the sand filters on an as-required basis. Nitric acid has been selected as the acid media due to its use in a similar application in industry.

In this study it is assumed that each filter is backwashed with dilute nitric acid (4%) every six months but actual practice could prove that less frequent acid treatment is necessary. The spent nitric acid is discharged to the tailings basin via the backwash surge pumps. It is assumed that the minimal flow of spent acid relative to the total flow to the tailings area will result in little effect on pH or nitrate values in the tailings basin. Provision for the neutralization of the spent acid is therefore not included.

### 5.4.4 Reagents

The reagent make-up facilities are shown on Drawing No. 100-10-8 with the lime circuit layout depicted on Drawing No. 100-10-9. The decant treatment reagent make-up layout is shown on Drawing No. 100-10-10.

Lime is received by truck and pneumatically unloaded to a 600 t lime silo. Lime slaking is carried out in a ball mill fed by a screw conveyor located beneath the lime silo. The mill discharge, slaked lime, is pumped to one of two lime slurry storage tanks from which the lime is pumped to tailings neutralization or other process points. The total on-site lime storage is sufficient for approximately two weeks of total process requirements.

5.4.4 (Cont'd)

Barium chloride, received in bags, is stored at the decant treatment building. It is manually made up as a 5% solution in a small mix tank from which it is transferred to a storage tank for metering to the first decant treatment tank for precipitating radium.

Flocculant is made up as a 0.05% solution at the decant treatment building using decant water for make-up. It is metered from the mix tank directly to the second decant treatment tank to effect flocculation of the suspended solids.

Nitric acid is received by tank truck as 65-70% $HNO_3$ and unloaded to a stainless steel holding tank located at the decant treatment building. An in-line mixer is used to dilute the acid to 4% with decant water for use in acid treating the sand filters.

A 1,000 $m^3/h$ process air compressor supplies air for lime agitation and other miscellaneous agitation requirements. Standby air and instrument air is assumed available from the main process air supply. A small 200 $m^3/h$ air compressor is included at the decant treatment building for air scour of the sand filters in the backwash cycle.

Process water requirements for waste treatment are minimal and water requirement for reagent make-up is assumed available from the main process plant supply.

## 5.5 TAILINGS CONTAINMENT

Model B tailings basin is constructed on a glacial till base with negligible topographic relief. Surface features and vegetation are similar to the eastern Athabasca Basin of northern Saskatchewan. The area is void of bedrock outcrops. A site plan and details of the basin and components are shown on Drawing Nos. 100-35-3 and 100-35-4.

Model B tailings are pumped from the mill to the tailings basin via two horizontal centrifugal pumps in series and a polyethylene tailings line. Open-end tailings discharge is practiced. As with Model A, the pump and pipeline systems are duplicated to ensure uninterrupted service.

The tailings lines are installed on a bench adjacent to the tailings access road. No lateral or vertical supports are included for restraining the pipe. The discharge ends are free and thermal expansion or contraction is not expected to cause large stresses. The tailings lines extend along both sides of the tailings basin and can be broken at each flange connection to allow discharge of tailings in various areas of the tailings basin. This allows a reduction in the effects of the natural angle of repose of tailings and hence maintains a more horizontal basin surface. Tailings are expected to settle at approximately a 1 degree angle of repose.

The tailings basin is constructed on glacial till after clearing of brush and excavation of unsuitable material of about one metre. Clearing would also be required in borrow areas and for all other structures. After excavation of unsuitable material, the base of the tailings basin would be treated with bentonite to a depth of approximately 300 mm and compacted to form the basin seal. A 1,500

## 5.5 (Cont'd)

mm till-bentonite seal would also be constructed on the interior dyke faces.

The tailings containment basin is constructed on a flat section of land and has length to width dimensions of 1,300 m by 650 m. Dyke construction is staged with initial starter dykes 3.5 m high and bi-annual raising of the dyke by 1.3 m. The dyke sections extend around the entire basin and are constructed of till material excavated adjacent to the site. Both interior and exterior slopes are protected from erosion. Staged dyke construction has been assumed in this model for minimizing front-end capital requirements.

Beginning at approximately the 7 meter level, a drain of sand and gravel would be included in the downstream dyke toe. The drain would be extended to the ultimate exterior toe as construction progressed. Seepage would be collected from the drain by a series of drainage pipes and manholes and monitored to check seal performance. A series of piezometers are installed around the perimeter of the basin to monitor groundwater conditions.

Excess water from the tailings basin is pumped from a floating barge in the pond to the decant treatment building. The water to be treated includes water contained in the tailings slurry, net rainfall and sand filter backwash water.

Other features constructed in the tailings area include the following:

- 1.1 kilometre access road from the mill to the tailings pond

**5.5 (Cont'd)**

- 2.5 kilometre, 3 phase, overhead power supply line and substation at the decant treatment building
- provision for flow measurement in tailings lines at the mill and basin to detect line leaks
- decant line from pond to decant treatment building
- sludge return line from treatment building to pond.

## 5.6 MONITORING REQUIREMENTS

The environmental monitoring requirements for Model B have been detailed according to Saskatchewan Environment and AECB regulations. The sample locations, the constituent analysis, the frequency of sampling and the types of samples are listed in Table C-3 in Appendix C.

The sample stations for liquid samples in Model B include:

- sand filter discharge (1)
- monitoring wells (4)
- toe drains (3)
- groundwater discharges (2)
- discharge stream (1)
- downstream stations (2)
- downstream lakes (7)
- mine area drainage ditch (1)
- mine area drainage (1)
- downstream from mine area (2)
- downstream bay (1).

In addition, a total of 11 stations have been assumed for sediment sampling as well as 4 stations for biota sampling and analysis.

A summary of analytical requirements (Tables C-4 and C-5) indicates a total of 5,270 determinations are required on an annual basis. It is assumed that on-site analysis is carried out using the assay office which services the whole operation.

## 5.7 CAPITAL COST ESTIMATES

### 5.7.1 Summary

The capital cost summary presented in Table 5.1 provides the estimated pre-production costs for the Model B waste management system for a 1,512 t/d ore processing facility. These estimated costs total $25,445,000 in first quarter 1984 Canadian dollars and provide for all pre-production costs of the waste management aspects of both minewater and mill tailings treatment.

In addition to this pre-production capital cost sustaining capital is required for Model B due to the nature and design of the tailings impoundment area which includes a gradual increase in storage capacity of the tailings basin throughout the life of the operation. For purpose of this study, it has been assumed that the height of the dykes around the tailings impoundment area will be increased on a bi-annual basis. This staged construction of the dykes is indicated on Drawing No. 100-35-4.

In order to accommodate the expenditures of this staged construction, a sustaining capital requirement of $20,200,000 has been included through the life of the operation. This bi-annual expenditure, complete with the height of the dykes, is identified in Table 5.2. The expenditures are based on increasing the height of the dykes by 1.25 metres every two years. As the total dyke height increases, a larger volume of earthwork is required to achieve this increase in height which leads to an increase in the bi-annual sustaining capital requirement.

The capital cost estimates include:

- building structures
- minewater handling circuit

## TABLE 5.1

## MODEL B
## CAPITAL COST ESTIMATE

| | |
|---|---|
| **Direct Costs** | |
| Minewater Treatment | $ 266,000 |
| Mill Effluent Treatment | 2,511,000 |
| Tailings Line | 308,000 |
| Tailings Storage Basin | 13,301,000 |
| Tailings Decant Treatment | 2,388,000 |
| Monitoring | 282,000 |
| Total Direct Costs | $ 19,056,000 |
| **Indirect Costs** | |
| Construction Indirects (8%) | $ 1,525,000 |
| Engineering, Procurement and Construction Management (12%) | 2,287,000 |
| Capital Spares | 264,000 |
| Subtotal | 23,132,000 |
| Contingency (10%) | 2,313,000 |
| Subtotal (Model B Pre-production Capital Costs) | 25,445,000 |
| Sustaining Capital (Tailings Storage Basin) | 20,200,000 |
| TOTAL MODEL B CAPITAL COSTS | $ 45,645,000 |

## TABLE 5.2

### MODEL B
### SUSTAINING CAPITAL COST ESTIMATE
### (1984 DOLLARS)

| Year | Height of Dyke (metres) | Dollars |
|---|---|---|
| 1984 | 3.50 | - |
| 1985 | 4.75 | 1,000,000 |
| 1987 | 6.00 | 1,200,000 |
| 1989 | 7.25 | 2,100,000 (1) |
| 1991 | 8.50 | 1,900,000 |
| 1993 | 9.75 | 2,200,000 |
| 1995 | 11.00 | 2,600,000 (2) |
| 1997 | 12.25 | 2,500,000 |
| 1999 | 13.50 | 2,600,000 |
| 2001 | 15.00 | 4,100,000 |
| TOTAL | | 20,200,000 |

NOTES

(1) Includes the material for the initial dyke seepage collection system.

(2) Includes a sand and gravel shell which serves as a future internal drain.

**5.7.1 (Cont'd)**

- mill tailings treatment circuit including reagent make-up facilities
- proportioned costs of a lime circuit
- tailings lines
- tailings storage basin
- tailings decant treatment
- monitoring
- indirect costs of construction
- engineering, procurement, and construction management
- contingency allowance.

The detailed pre-production capital cost estimate for Model B is provided under separate cover.

A breakdown of the capital costs for each major area is presented in Table 5.3. These costs are based on the total pre-production capital costs with indirect capital costs added to the direct capital costs of each area on a proportioned basis. The sustaining capital cost for the tailings basin is shown as a separate item and is part of the total capital costs for Model B. Unit capital costs relative to the annual uranium production and annual tonnage throughput of the facility are also presented by area.

On this basis, the total unit capital costs for Model B are:

- $90.82 per annual tonne of ore processed
- $23.73 per annual kilogram of U produced
- $9.13 per annual pound of $U_3O_8$ produced.

These cost estimates do not include any provision for pre-production construction interest as the total value of this interest is dependent on the method of financing and the equity or credit

## TABLE 5.3

## MODEL B
## CAPITAL COSTS BY AREA

| Area | Dollars | $/annual tonne | $/annual kg U | $/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 355,200 | 0.71 | 0.19 | 0.07 |
| Mill Effluent Treatment | 3,352,900 | 6.67 | 1.74 | 0.67 |
| Tailings Line | 411,300 | 0.82 | 0.21 | 0.08 |
| Tailings Storage Basin | 17,760,500 | 35.34 | 9.23 | 3.55 |
| Tailings Decant Treatment | 3,188,600 | 6.34 | 1.66 | 0.64 |
| Monitoring | 376,500 | 0.75 | 0.20 | 0.08 |
| Subtotal (Pre-production) | 25,445,000 | 50.63 | 13.23 | 5.09 |
| Sustaining Capital (Tailings Storage Basin) | 20,200,000 | 40.19 | 10.50 | 4.04 |
| TOTAL | 45,645,000 | 90.82 | 23.73 | 9.13 |

**5.7.1 (Cont'd)**

position of the specific investor. These pre-production interest costs can have a significant impact on the total costs of any project and this should be kept in mind when reviewing the cost estimates given in this report.

## 5.7.2 Basis of Estimate

The capital cost estimate is based on the following information:

- flowsheets and conceptual layouts prepared by Kilborn and Melis Consulting Engineers Ltd.
- process and material specifications as supplied by Melis Consulting Engineers Ltd.
- estimated construction material quantities as derived by Kilborn
- budget quotations on major equipment
- pricing data from Kilborn's files and records.

For purposes of this study, it has been assumed that:

- all construction services would be tendered by fixed price contract. The average labour rate is $50.00 per hour based on a 60-hour work week. This labour rate includes provision for direct wages, payroll burdens and benefits, small tools and consumables, site supervision, subsistence, travel time and expenses, and the contractor's overhead and fee

- construction labour would be transported to site and work a three-week in/one-week out schedule and reside in a construction camp while on site

## 5.7.2 (Cont'd)

- all costs are based on first quarter 1984 Canadian dollars
- the cumulative price factor, applied to base prices for process equipment is nineteen percent. This includes:
    - provincial sales tax - 5 percent
    - freight allowance - 8 percent
    - miscellaneous minor materials allowance - 5 percent

No allowances have been made in the capital estimate for:

- escalation beyond the first quarter 1984
- preproduction construction interest
- working capital
- production equipment or materials
- purchase of land or right of way easements
- start-up assistance
- cost of pre-appropriation studies, development work or environmental studies and hearings
- owners' costs such as an owners' representative
- general property facilities such as housing and administration offices for the operations personnel
- pumping minewater from the mine to the clarification pond
- decommissioning.

## 5.7.3 Methodology

Equipment prices were obtained on an individual basis from budget quotations or historical data. Mechanical installation costs, including field material, were factored from the equipment costs.

### 5.7.3 (Cont'd)

In-plant piping, electrical and instrumentation costs were factored from the total installed mechanical costs. Building structure costs for each area were based on volumes and volume unit prices derived from historical data. The building costs includes the cost of the buildings for waste treatment facilities as well as tailings decant treatment.

The lime circuit is sized and priced for the total mill requirements including both production and waste management. Sixty-seven percent of this total cost was proportioned to the waste management area based on the percentage lime usage for waste management.

The monitoring cost includes piezometers and sampling wells for the tailings area. It is assumed that thirty percent of analytical efforts for a uranium operation are dedicated to environmental requirements, therefore this portion of the capital cost of the laboratory building and equipment is included the monitoring cost.

The allowance for construction indirects has been estimated as eight percent of the total direct costs. It includes:

- temporary site services
- permits, insurance and bonds
- winter works
- rental of mobile equipment
- on site construction supervision facilities
- pre-construction geotechnical and hydrogeotechnical investigation.

### 5.7.3 (Cont'd)

The engineering, procurement and construction management function has been estimated as twelve percent of the direct costs. As well as design engineering, this includes provision for project management, cost control, scheduling, contract administration and inspection services.

An allowance of five percent of home office costs has been included for capital spares inventory. The contingency allowance provides for additions and unidentified items within the stated scope of work but it is not intended to cover major changes in scope.

## 5.8 OPERATING COSTS

### 5.8.1 Summary

The total waste management operating cost for Model B, estimated from the design basis given in section 5.3, is $4,357,372 annually in 1984 dollars, including a five percent contingency. This cost is equivalent to $8.67/tonne of ore processed or $2.27/kg U ($0.87/lb $U_3O_8$) based on an annual uranium production of 1,923,000 kg of U (5,000,000 lbs $U_3O_8$).

The Model B operating costs are summarized in Table 5.4 and an estimated breakdown of the costs for each area of the waste management system is provided in Table 5.5. Cost details are given in Tables 5.6, 5.7 and 5.8.

The operating cost breakdown given in Table 5.5 is a proportioned estimate of the total cost allocated to the following areas:

- minewater treatment
- mill effluent treatment
- tailings line
- tailings basin
- tailings decant treatment
- monitoring

The costs of reagents and supplies have been proportioned according to the estimated usage for each area.

The minewater treatment operating costs include the minimal costs associated with clarification of the minewater. The mill effluent treatment costs include all associated costs of treating mill waste

## TABLE 5.4

## MODEL B
## OPERATING COST SUMMARY

| Item | Annual $ | $/tonne | $/kg U | $/lb $U_3O_8$ |
|---|---|---|---|---|
| Manpower | 657,695 | 1.31 | 0.34 | 0.13 |
| Waste Treatment | 3,372,183 | 6.71 | 1.75 | 0.67 |
| Roads & Yards | 20,000 | 0.04 | 0.01 | - |
| Accommodation | 70,000 | 0.14 | 0.04 | 0.02 |
| Transportation | 30,000 | 0.06 | 0.02 | 0.01 |
| Contingency (5%) | 207,494 | 0.41 | 0.11 | 0.04 |
| TOTAL | 4,357,372 | 8.67 | 2.27 | 0.87 |

## TABLE 5.5

## MODEL B
## OPERATING COSTS BY AREA ($)

| Item | Minewater Treatment | Mill Effluent Treatment | Tailings Line | Tailings Basin | Tailings Decant Treatment | Monitoring | Total |
|---|---|---|---|---|---|---|---|
| Manpower | 18,664 | 186,641 | 37,328 | 18,664 | 111,985 | 284,413 | 657,695 |
| Reagents & Supplies | 6,250 | 2,324,232 | 12,500 | 6,250 | 132,482 | 10,000 | 2,491,714 |
| Power | 32,438 | 593,045 | - | - | 182,463 | 3,000 | 810,946 |
| Heating | - | 49,459 | - | - | 18,564 | 1,500 | 69,523 |
| Roads & Yards | 2,500 | - | 5,000 | 10,000 | 2,500 | - | 20,000 |
| Accomodation & Transportation | 2,800 | 28,400 | 5,700 | 2,800 | 17,000 | 43,300 | 100,000 |
| SUB-TOTAL | 62,652 | 3,181,777 | 60,528 | 37,714 | 464,994 | 342,213 | 4,149,878 |
| Contingency (5%) | 3,133 | 159,088 | 3,026 | 1,886 | 23,250 | 17,111 | 207,494 |
| TOTAL | 65,785 | 3,340,865 | 63,554 | 39,600 | 488,244 | 359,324 | 4,357,372 |
| $/tonne | 0.13 | 6.65 | 0.13 | 0.08 | 0.97 | 0.71 | 8.67 |
| $/kg U | 0.03 | 1.74 | 0.03 | 0.02 | 0.26 | 0.19 | 2.27 |
| $/lb $U_3O_8$ | 0.01 | 0.67 | 0.01 | 0.01 | 0.10 | 0.07 | 0.87 |

## TABLE 5.6

## MODEL B
## ANNUAL MANPOWER COSTS

| Employee | Requirement | Number | Rate | Annual $ |
|---|---|---|---|---|
| Supervision: | | | | |
| - Operational | 0.2 man/shift | 0.8 | $3,675/month | 35,280 |
| - Maintenance | 0.2 man/day | 0.4 | $3,675/month | 17,640 |
| - Environmental | 0.5 man/day | 1.0 | $3,675/month | 44,100 |
| - Analytical | 0.2 man/day | 0.4 | $3,415/month | 16,390 |
| - Process Engineer | 0.1 man/day | 0.2 | $3,415/month | 8,195 |
| Environmental Sampler | 1.0 man/day | 2.0 | $2,625/month | 63,000 |
| Analytical Technician | 1.5 men/day | 3.0 | $2,890/month | 104,040 |
| Waste Treatment Operator | 1.0 man/shift | 4.0 | $15.75/hour | 121,275 |
| Tails Area Operator | 0.4 man/shift | 1.6 | $14.70/hour | 45,275 |
| Waste Treatment Area Maintenance | 20 mandays/month | 1.3 | $16.80/hour | 44,350 |
| Tails Area Maintenance | 12 mandays/month | 0.8 | $16.80/hour | 26,610 |
| SUB-TOTAL | | 15.5 | | 526,155 |
| Payroll Burden (25%) | | | | 131,540 |
| TOTAL | | | | 657,695 |

## TABLE 5.7

## MODEL B
## WASTE TREATMENT COSTS

| Item | Annual $ | $/tonne | $/kg U | $/lb $U_3O_8$ |
|---|---|---|---|---|
| Reagents & Supplies | 2,491,714 | 4.96 | 1.30 | 0.50 |
| Power | 810,946 | 1.61 | 0.42 | 0.16 |
| Heating | 69,523 | 0.14 | 0.03 | 0.01 |
| TOTAL | 3,372,183 | 6.71 | 1.75 | 0.67 |

## TABLE 5.8

## MODEL B
## REAGENTS AND SUPPLIES

| Item | Unit Cost ($/t) | Consumption kg/t* | Consumption t/a | Annual $ |
|---|---|---|---|---|
| Lime (95% CaO) | 150 | 30.0 | 15,078 | 2,261,732 |
| Barium Chloride | 411 | 0.26 | 132.1 | 54,293 |
| Flocculant | 4,500 | 0.005 | 2.642 | 11,889 |
| Nitric Acid | 576 | 0.1 | 50 | 28,800 |
| Laboratory Supplies | | | | 10,000 |
| Maintenance Supplies | | | | 100,000 |
| Operating Supplies | | | | 25,000 |
| TOTAL | | | | 2,491,714 |

* Reagent consumptions are based on an annual mill feed of 502,607 tonnes.

## 5.8.1 (Cont'd)

solutions. The decant treatment costs include the cost of treating tailings decant water at the tailings area. Monitoring costs are the costs associated with environmental sampling and analysis. The tailings line and tailings basin costs are the operating costs associated with these two areas.

## 5.8.2 Manpower

The manpower costs detailed in Table 5.6 are those required to operate the waste treatment circuits and tailings area as well as meeting monitoring requirements for the facility. The annual total of $657,695 includes a twenty-five percent allowance for payroll burden.

The indicated salary rates have been estimated by increasing the salary and union rates used in Model A by five percent to reflect a higher rate for the fly-in, fly-out schedule worked at this facility. The total number of employees required is double the manpower allowance since a 7-day in, 7-day out schedule necessitates the doubling of manpower to have full year-round coverage.

Operational supervision consists of twenty percent of a supervisor's time on each shift to cover waste management related areas. The supervision allowance for maintenance, environmental, analytical and a process engineer is restricted to dayshift only.

One environmental sampler spends full time on sampling and field measurements to meet the sampling requirements for Model B as detailed in Appendix C.

### 5.8.2 (Cont'd)

The analytical workload for Model B is similar to Model A with an allowance of 1.5 analytical technicians working on environmental analysis on a daily basis.

A total of 1.4 men per shift has been allowed for operating the waste treatment circuits, monitoring the tailings area and making up reagents as required. The maintenance allowance for all areas is 32 mandays per month. Hourly employees work a schedule of 2 shifts per day, 11 hours per shift, 350 days per year.

### 5.8.3 Reagents and Supplies

The annual cost of reagents and supplies ($2,491,714) is based on the consumption figures given in the design basis and the delivered price of reagents at the Model B site. The higher lime cost reflects the higher transportation cost to northern Saskatchewan relative to the transportation costs for the Elliot Lake area in Model A. The annual reagent requirements are based on annual tonnage of 502,607 tonnes of ore as mill feed.

The lime cost only includes the requirements for tailings neutralization. Lime requirements for process have not been included.

Barium chloride, flocculant and nitric acid consumptions are the annual requirements for the decant treatment facility operating 365 days per year. They have been calculated for the decant water flow being treated but are expressed as kg/t of ore processed.

## 5.8.3 (Cont'd)

Lump sum estimates have been used for laboratory supplies, maintenance supplies and general operating supplies. These estimates are only to meet the cost of the waste management section of the facility.

The consumption of sulphuric acid for pH adjustment has been assumed as negligible and is not included in the costs.

## 5.8.4 Power

The total power requirements for the waste management section of the Model B facility is estimated at 746 kW (1,000 HP). The cost of diesel-generated power is assumed at $0.11/kWh. Based on an 85% power efficiency factor and a 350 days per year operation, the annual power cost is $810,946.

## 5.8.5 Heating

An allowance has been included for heating the waste treatment building, the decant treatment building and a portion of the site laboratory building. Annual propane requirements are estimated at 331,060 L at a cost of $0.21/L for a total heating cost of $69,523. This includes additional heating to allow for sufficient air changes in the waste treatment building that may be required for radon gas control.

### 5.8.6 Miscellaneous

A lump sum allowance of $20,000 has been used for the cost of maintaining roads and yards associated with the waste management section of the Model B facility.

On-site accommodation costs have been calculated at $25.00 per bed per day for the total manpower given in Table 5.4. A total of 15.5 employees would require 7.8 beds to provide the accommodation for waste management personnel. This calculates to a figure of approximately $70,000 per year for company-sponsored room and board costs.

Employee transportation costs of $30,000 per year have been estimated for 15.5 employees with 23 round trips per year per employee at a rate of $80 per round trip.

## 5.9 CASH FLOW AND NET PRESENT VALUE

A net cash flow for the Model B waste management system is presented in Table 5.9. This cash flow is based on the construction schedule indicated in the terms of reference for this study. The key dates are:

- Engineering Start: January 1980
- Construction Start: June 1981
- Plant Startup: August 1983
- Plant decommissioning: 2004

It has also been assumed that no significant production would occur during the period August 1983 to December 1983 and there would therefore be no significant operating costs during this period.

Normally a total cash flow is the summation of revenue, operating costs, interest expenses, freight and transportation, fixed capital, working capital, sustaining capital, federal taxation, provincial taxation and selling expenses.

Since this study presents a net cash flow for waste management only, the net cash flow is defined as the total project cash flow including the waste management costs minus the total project cash flow excluding the waste management costs. Therefore, in the net cash flow calculation, all costs that are not affected by waste management, such as revenue, freight, transportation and selling expenses, can be excluded from consideration.

**5.9 (Cont'd)**

Since the study criteria also excluded taxation considerations federal and provincial taxes have been excluded in the calculations. In addition, since interest expenses are dependent on financing and are investor-specific they are also excluded in this analysis.

Consequently, the net cash flow equation used for calculating the incremental waste management costs is:

Cash Flow = operating costs + fixed capital + working capital + sustaining capital.

A net present value of minus $46,540,000 has been determined for Model B based on the cost elements of the net cash flow given in Table 5.9. This net present value is based on a discount rate of 12.5% and gives the net present value in 1984 constant dollars at the date in time when the initial capital is committed (i.e. 1980). Since this calculation is on a pre-tax basis, no consideration is given for capital cost allowance or depreciation.

## TABLE 5.9

### MODEL B
### CASH FLOW
(CONSTANT 1984 $ x 1,000,000)

| | 1980 | 1981 | 1982 | 1983 | 1984 | 1985 | 1986 | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | 2000 | 2001 | 2002 | 2003 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 |
| Operating Cost | | | | | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 | 4.4 |
| Capital Cost: | | | | | | | | | | | | | | | | | | | | | | | | |
| - initial | 0.6 | 5.0 | 10.0 | 9.8 | | | | | | | | | | | | | | | | | | | | |
| - sustaining | | | | | | 1.0 | | 1.2 | | 2.1 | | 1.9 | | 2.2 | | 2.6 | | 2.5 | | 2.6 | | 4.1 | | |
| Working Capital | | | | | 0.3 | | | | | | | | | | | | | | | | | | | |
| Total Costs | 0.6 | 5.0 | 10.0 | 10.1 | 4.4 | 5.4 | 4.4 | 5.6 | 4.4 | 6.5 | 4.4 | 6.3 | 4.4 | 6.6 | 4.4 | 7.0 | 4.4 | 6.9 | 4.4 | 7.0 | 4.4 | 8.5 | 4.4 | 4.4 |
| Present Value Factor | 1.0 | 0.89 | 0.79 | 0.70 | 0.62 | 0.55 | 0.49 | 0.44 | 0.39 | 0.35 | 0.31 | 0.27 | 0.24 | 0.22 | 0.19 | 0.17 | 0.15 | 0.14 | 0.12 | 0.11 | 0.09 | 0.08 | 0.07 | 0.07 |
| Present Value | 0.6 | 4.45 | 7.90 | 7.07 | 2.73 | 2.97 | 2.16 | 2.46 | 1.72 | 2.28 | 1.36 | 1.70 | 1.06 | 1.45 | 0.84 | 1.19 | 0.66 | 0.97 | 0.53 | 0.77 | 0.40 | 0.68 | 0.31 | 0.31 |
| Net Present Value | 0.6 | 5.05 | 12.95 | 20.02 | 22.75 | 25.72 | 27.88 | 30.34 | 32.05 | 34.33 | 35.69 | 37.39 | 38.45 | 39.90 | 40.74 | 41.93 | 42.59 | 43.55 | 44.08 | 44.85 | 45.25 | 45.93 | 46.23 | 46.54 |

# 6.0 MODEL C STUDY

## 6.1 INTRODUCTION

The Model C uranium property is an open-pit mine located in the Athabasca Basin area of northern Saskatchewan. The ore being treated is a high-grade uranium ore containing arsenic and heavy metal mineralization.

The annual uranium production of 1,923,000 kg U (5,000,000 lbs $U_3O_8$) is based on an annual mill tonnage of 155,078 tonnes and a 97.5% uranium recovery. The mill throughput is 466.4 tonnes per day based on a 350 days per year operation and 95% plant availability.

Separate neutralization and treatment of mill waste solutions is practiced for removal of radium and other contaminants with clarification of treated effluent by sand filtration. The waste sludge is dewatered by thickening and filtration and combined with the neutralized process tailings for disposal to the tailings area. Minewater is clarified and used as process water with excess water being treated in a separate circuit for radium, arsenic and heavy metal removal. The waste precipitates are collected in settling ponds and treated water is clarified by sand filtration. All treated effluent report to monitoring ponds prior to discharge to the environment.

The tailings containment area utilizes the layered tailings concept for deposition which provides for partially-dried tailings suitable for immediate decommissioning at mine closure. Seepage and runoff from the tailings area is returned to the mill for treatment in the mill waste solution treatment circuit.

**6.1 (Cont'd)**

It has been assumed that there is an all-weather road to the property but the site is remote and requires diesel generated power. As in Model B, a 7-day in, 7-day out schedule is worked by all site employees and company-sponsored accommodation and air transportation from established communities is provided for the employees.

## 6.2 GENERAL PROCESS DESCRIPTION

The mineralogy of the ore for Model C consists of pitchblende uranium mineralization along with gersdorffite, millerite, kaolinite, chlorite, and quartzite.

The general process flowsheet for Model C is depicted on Drawing No. 100-10-11. The circuit includes grinding, neutral pulp storage, pressure leaching, liquid/solid separation using CCD thickeners, solvent extraction and ammonia precipitation of yellowcake. Separate waste treatment circuits are provided for liquid waste, solid tailings and minewater.

Grinding is carried out to 85% passing 250 microns prior to acid leaching the ore in pressure autoclaves under oxygen pressure at a temperature of 70°C and a sulphuric acid consumption of 120 kg/t. Eight stages of CCD thickeners are used for recovering the pregnant solution and washing the leached residue. Wash solution to the CCD circuit is acidified process water.

The pregnant solution is clarified in pressure filters prior to reporting to the solvent extraction circuit for uranium recovery. The loaded organic is stripped of uranium using ammonium sulphate solution to produce the pregnant strip liquor which is precipitated with ammonia to recover the uranium as yellowcake product. The raffinate is discarded to the waste treatment circuit along with spent sodium carbonate solution from organic regeneration.

Barren strip from the yellowcake precipitating circuit is clarified and recycled as ammonium sulphate strip solution in the solvent extraction circuit. A bleed stream of barren strip reports to an ammonium sulphate crystallization plant where ammonium sulphate

## 6.2 (Cont'd)

crystals are recovered for achieving acceptable ammonia levels in effluent streams.

The raffinate and spent sodium carbonate solutions are treated for contaminant removal by neutralization with lime and precipitation with barium chloride. The clarified treated effluent from the circuit reports to monitoring ponds prior to release to the environment. Waste precipitates are dewatered and combined with the CCD tailings for bulk neutralization with lime and disposal to the layered tailings area. Seepage and runoff from the tailings area is recycled to the raffinate neutralization circuit for treatment.

Minewater, containing heavy metals, arsenic and radium contaminants, is clarified in a pond and used as process water. Excess water is treated in a separate circuit with lime, ferric sulphate and barium chloride for removal of contaminants. The treated, clarified minewater reports to the monitoring ponds prior to discharge to the environment.

## 6.3 MODEL C CRITERIA

The design criteria used in the Model C study is based on the original design basis detailed in Appendix B with minor revisions being made to reflect a similar type of operation in northern Saskatchewan. The Model C design basis is given below.

### 6.3.1 General

| | |
|---|---|
| Annual uranium production | - 1,923,000 kg U (5,000,000 lbs $U_3O_8$) |
| Total years of operation | - 20 |
| Operating days per year | - 350 |
| Plant availability | - 95% |
| Net uranium recovery | - 97.5% |
| Mill throughput (t/d) | - 466.4 |
| Mill throughput (t/h) | - 19.4 |
| Annual mill tonnage (tonnes) | - 155,078 |
| Mill feed grade (% $U_3O_8$) | - 1.5 |

### 6.3.2 Process

| | |
|---|---|
| Ore grind | - 85% passing 250 microns |
| Leach sulphuric acid consumption (kg/t) | - 120 |
| Weight loss in leach | - 10% |
| Ore specific gravity | - 2.8 |
| Leach liquor specific gravity | - 1.11 |
| Pregnant solution specific gravity | - 1.05 |
| Pregnant strip liquor specific gravity | - 1.08 |
| Raffinate specific gravity | - 1.02 |
| Barren strip liquor specific gravity | - 1.08 |
| Leach discharge density (% solids) | - 43 |
| CCD tails density (% solids) | - 45 |

### 6.3.2 (Cont'd)

| | |
|---|---|
| Pregnant solution/ore ratio (w/w) | - 4.25/1 |
| Pregnant strip liquor flow ($m^3/h$) | - 14.3 |
| Barren strip flow ($m^3/h$) | - 19.2 |
| Barren strip bleed to crystallization ($m^3/h$) | - 5.6 |
| Organic regeneration | - 20% of flow |

### 6.3.3 Waste Treatment

### 6.3.3.1 Minewater

Flows to minewater pond - minewater
- mill site runoff
- stockpile runoff
- equipment wash

| | |
|---|---|
| Minewater flow ($m^3/h$) | - 144 |
| Miscellaneous contaminated water ($m^3/h$) | - 10 |
| Minewater flow to process ($m^3/h$) | - 88.8 |
| Minewater flow to treatment ($m^3/h$) | - 65.2 |
| Suspended solids (mg/L) | - 260 |
| Minewater pH | - 6 |
| Lime consumption ($kg/m^3$) | |
| - first stage | - 0.021 |
| - second stage | - 1.0 |
| Barium chloride addition ($kg/m^3$) | - 0.1 |
| Ferric sulphate addition (Ferri-Floc) ($kg/m^3$) | - 3 |
| Flocculant addition ($g/m^3$) | |
| - first stage | - 2 |
| - second stage | - 2 |

### 6.3.3.1 (Cont'd)

Total treatment retention time (h) - 4
Stages - 2
Precipitation pH - 8
Final neutralization pH - 11
Clarification pond retention time (h) - 6
No. 1 settling pond retention time (h) - 10
No. 2 settling pond retention time (h) - 48
Sand filter unit flowrate ($m^3/h/m^2$) - 15
Number of sand filters - 2 operating, plus 1 standby or backwash
Backwash flow - 4% of feed volume
Nitric acid wash - semi-annual
Clarification pond slimes estimate (t/a) - 315
No. 1 settling pond sludge estimate (t/a) - 335
No. 2 settling pond sludge estimate (t/a) - 1,165
Net discharge to monitoring ponds ($m^3/h$) - 69
Discharge pH - 7
Minewater slimes to process - annually
Minewater precipitate sludge to tailings basin - annually

### 6.3.3.2 Raffinate Treatment

Treatment streams - raffinate
- spent sodium carbonate solution
- seepage and runoff from tailings pond
- sumpage

Number of stages - 2
Retention time per stage (h) - 2
Neutralization pH - first stage - 8
- second stage - 11
Lime consumption (kg/$m^3$)
- first stage - 16
- second stage - 1.6
Barium chloride addition (kg/$m^3$) - 0.1
Flocculant addition per stage (g/$m^3$) - 2
First stage precipitate (t/d) - 144
Second stage precipitate (t/d) - 12
Clarifying thickener rise rate ($m^3$/h/$m^2$) - 2.5
Drum filtration unit rate (t/h/$m^2$) - 0.18
No. 1 clarifying thickener underflow density (% solids) - 15
No. 2 clarifying thickener underflow density (% solids) - 5
Drum filter cake density (% solids) - 25
Sand filtration unit flowrate ($m^3$/h/$m^2$) - 15
Number of sand filters - 2 operating, plus 1 standby or backwash
Backwash flow - 4% of feed volume
Nitric acid wash - semi-annual
Net discharge to monitoring ponds ($m^3$/h) - 124.9
Discharge pH - 7
Monitoring pond retention time (h) - 48

### 6.3.3.3 Tailings Neutralization

Treatment streams - CCD tails
- No. 2 clarifying thickener underflow
- drum filter cake discharge

Neutralization retention time (h) - 8
Neutralization pH - 11
Lime consumption (kg/t) - 40
Final tailings density (% solids) - 33
Tailings flow (t/h) - 25.0
Surge capacity (h) - 72

### 6.3.3.4 Ammonium Sulphate Crystallization

Barren strip bleed flow ($m^3/h$) - 5.6
Ammonium sulphate production (kg/kg $U_3O_8$)- 2.5
Ammonium sulphate production (t/h) - 0.67
Ammonium sulphate production (t/year) - 5,700
Steam requirement (t/h) - 1.8
Propane for dryer (L/h) - 12.5
Cooling tower water flow ($m^3/h$) - 165

### 6.3.3.5 Tailings Basin

Deposition method - layered tailings
Design life - 20 years with complete construction prior to mill start-up
Maximum rainfall and snowmelt events controlled within basin
Exterior dyke slopes - 2.74:1

**6.3.3.5 (Cont'd)**

Interior dyke slopes on upstream and lateral embankments - 2:1

Interior dyke slopes on downstream embankment - 4:1

Top width of dyke (m) - 8

Minimum freeboard (m) - 2

Average surface area of tailings (ha) - 42.6

Catchment area (ha) - 49

Average final tailings depth (m) - 13

Flow to tailings basin

- process tailings (t/h) - 25.0
- minewater sludge (t/a) - 1,500

Tailings solids specific gravity - 2.6

Minewater dry sludge specific gravity - 2.2

Minewater sludge density (% solids) - 5

Water in tailings slurry ($m^3$/h) - 50.9 (continuous)

Seepage and runoff flow ($m^3$/h) - 37

Net precipitation ($m^3$/h) - 1

Final dry density of tailings (t/$m^3$) - 0.85

Mean annual precipitation (mm) - 460

Mean annual evaporation (mm) - 508

Runoff coefficient for precipitation - 0.9

Water recycle line to tailings basin ($m^3$/h) - 250

### 6.3.3.6 **Minewater, Settling and Monitoring Ponds**

Storage, treatment and monitoring of all liquid waste for Model C includes six earthen ponds constructed adjacent to the mill building. These are designated as:

- minewater storage pond
- settling ponds 1 and 2
- monitoring ponds 1, 2 and 3.

The design basis for these ponds includes the following:

| | |
|---|---|
| Interior slopes | - 2.5:1 |
| Exterior slopes | - 3:1 |
| Minimum freeboard (m) | - 1.5 |
| Top of dyke width (m) | - 8 |
| Embankment fill material | - locally excavated till |
| Pond liners | - 100 mil (HDPE) |
| Pond seal and seepage collection | - perforated pipes below liner and till-bentonite seal below pipes |
| Overflow protection | - high level alarms<br>- overflow pipes to sump and pumped back to mill<br>- liners extended through embankments and down exterior dyke slopes to prevent washouts |

### 6.3.4 Reagents

#### 6.3.4.1 Lime

| | |
|---|---|
| Lime (95% CaO) consumption (kg/h) | |
| - raffinate neutralization | - 2,000 |
| - radium precipitation | - 200 |
| - tailings neutralization | - 600 |
| - minewater treatment | - 67 |
| - total | - 2,867 |
| Total lime consumption (kg/t) | - 155.3 |
| Lime slaking capacity (t/h) | - 5 |
| Lime storage (tonnes) | - 1,200 |
| lime slurry storage (days) | - 1 |
| Lime specific gravity | - 2.2 |
| Lime slurry density (% solids) | - 20 |
| Delivery | - bulk, dry |

#### 6.3.4.2 Barium Chloride

| | |
|---|---|
| Barium chloride addition (kg/h) | |
| - raffinate treatment | - 11.7 |
| - minewater treatment | - 6.5 |
| - total | - 18.2 |
| Total barium chloride consumption (kg/t) | - 0.99 |
| Solution concentration | - 5% |
| Delivery | - bags |

### 6.3.4.3 Ferric Sulphate

| | |
|---|---|
| Consumption (as Ferri-Floc) (kg/t) | - 10.2 |
| Solution strength | - 7% |
| Delivery | - bulk |

### 6.3.4.4 Flocculant

| | |
|---|---|
| Total flocculant consumption (kg/t) | - 0.04 |
| Solution strength | - 0.05% |
| Delivery | - bags |

### 6.3.4.5 Nitric Acid

| | |
|---|---|
| Delivered acid concentration | - 65 to 70% $HNO_3$ |
| Nitric acid solution concentration | - 4% |
| Nitric acid consumption (kg/t) | - 0.5 |
| Sand filter acid wash frequency | - semi-annual |
| Number of bed volumes per acid wash | - 4 |
| Delivery | - bulk |

## 6.4 WASTE TREATMENT DESCRIPTION

### 6.4.1 Minewater

The minewater treatment circuit is described on Drawing No. 100-10-14 and the conceptual layout of the equipment is indicated on Drawing No. 100-10-16. The minewater and settling ponds are shown on Drawing No. 100-35-7. In this study it is assumed that minewater is treated in a separate circuit.

Minewater from the open-pit operation is collected along with mill site and stockpile runoff and equipment wash in a synthetic-lined clarification pond with a minimum retention time of six hours. The suspended solids settle to the bottom of the pond and clarified minewater is pumped to the process water tank or to the minewater treatment circuit. The settled solids are removed annually by pumping to the mill grinding and dewatering circuit.

In the treatment circuit minewater is adjusted to pH 8 with lime in a mechanically-agitated tank followed by the addition of barium chloride in a second tank for precipitating radium. Flocculant is added in a flash-mix tank to coagulate the suspended solids prior to gravity discharge to a synthetic-lined settling pond of ten hours retention time for clarification of the treated water.

The clarified settling pond overflow is further treated in two mechanically-agitated tanks with ferric sulphate addition and lime to pH 11 for precipitating the remaining contaminants. Flocculant is added to the precipitate slurry in a flash-mix tank prior to gravity discharge to a second settling pond with a retention time of forty-eight hours.

### 6.4.1 (Cont'd)

The settled sludge from both settling ponds is pumped to the tailings area on an annual basis. An annual total of 1,500 dry tonnes of sludge from both settling ponds has been used in this study.

The overflow from the second settling pond is pumped to dual-media sand filters for final clarification prior to pH adjustment and discharge to the monitoring ponds at a rate of 69 $m^3/h$. Clarified effluent is used as backwash with the backwash solution containing trace amounts of solids reporting to No. 2 settling pond. A facility for acid wash of the media has been included with the spent nitric acid wash reporting to the second stage treatment circuit for neutralization.

### 6.4.2 Raffinate Treatment

The raffinate treatment circuit for Model C includes treatment of waste raffinate solution and spent sodium carbonate solution from the solvent extraction circuit, contaminated seepage and runoff water from the tailings basin and miscellaneous sumpage. The flowsheet is described on Drawing No. 100-10-13 and a conceptual layout of the equipment is shown on Drawing No. 100-10-16.

The combined waste solutions are neutralized with lime to pH 8 in a series of four air-agitated pachucas with a total of two hours retention time. Flocculant is added in a flash-mix tank and the precipitate slurry is dewatered in a high-rate type thickener to 15% solids. The underflow is dewatered further by drum filtration to 25% solids. The filtrate from the drum filters is returned to the thickener.

### 6.4.2 (Cont'd)

The overflow from the first stage treatment is treated with barium chloride for radium precipitation and then neutralized to pH 11 with lime to precipitate the remaining contaminants. This is accomplished in a series of four mechanically-agitated tanks with a total retention time of two hours. Flocculant is added in a flash-mix tank and the precipitate slurry is dewatered to 5% solids in a high-rate thickener. The underflow from this thickener is repulped with the drum filter cake discharge and the combined waste sludge is pumped to the tailings neutralization circuit for disposal to the tailings area with the neutralized CCD tailings.

The thickener overflow from second-stage treatment is clarified in dual-media sand filters prior to pH adjustment and discharge to the monitoring ponds at a rate of 124.9 $m^3/h$. Clarified effluent is used as backwash and a facility for nitric acid wash of the media is included. Both the backwash solution and the spent acid wash are returned to the second-stage treatment circuit.

There are a total of three monitoring ponds with a retention time of forty-eight hours for each pond. The size of each pond will allow sufficient time to carry out the required analysis for ensuring that the treated effluent meets environmental guidelines before discharging to the watershed. Treated effluent not meeting environmental guidelines is recycled for further treatment. The total flow to the monitoring ponds is 193.9 $m^3/h$ which includes treated raffinate and treated minewater. A layout and section of the monitoring ponds is shown on Drawing No. 100-35-7.

### 6.4.3 Tailings Neutralization

The tailings neutralization circuit is depicted on Drawing No. 100-10-13 and a conceptual layout of the equipment is shown on Drawing No. 100-10-16.

The washed leach residue from the CCD circuit and the waste sludge from the raffinate treatment circuit are bulk-neutralized with lime to pH 11 in two mechanically-agitated tanks with a total retention time of eight hours. A total of seventy-two hours surge capacity for the neutralized tailings is made available by a series of four tanks with mechanical agitation. This surge capacity would only be used for emergency reasons or for scheduled maintenance purposes.

The neutralized tailings slurry is pumped to the tailings basin at a density of 33% solids and at a rate of 25.0 t/h. Temperature control for periods of cold weather would be achieved by the addition of steam to the tailings pulp in the neutralization tanks. High-pressure glandless diaphragm pumps are used to pump the tailings slurry via a steel tailings line to the tailings basin. A standby pump and line are available for emergency or maintenance purposes.

### 6.4.4 Ammonium Sulphate Crystallization

The removal of ammonia from process streams is accomplished by the evaporation and crystallization of a bleed stream of clarified barren strip liquor to produce fertilizer-quality ammonium sulphate crystals.

As indicated on Drawing No. 100-10-15 this would be carried out in a package crystallization/dryer plant with steam supply originating from waste heat boilers recovering waste heat from other areas of

### 6.4.4 (Cont'd)

the processing facility. The ammonium sulphate crystals would be dried in a propane-fired dryer and transferred to a 600 t bin via a bucket elevator. The high-quality ammonium sulphate product would be trucked to market and sold as fertilizer.

### 6.4.5 Reagents and Utilities

The reagent make-up requirements are shown on Drawing No. 100-10-15 and the equipment layout is indicated on Drawing No. 100-10-16.

Lime is received in bulk by tank truck and pneumatically unloaded to two 600 t lime storage silos. Lime from each silo is fed to individual lime slakers with a total slaking capacity of 5 t/h. Lime slurry is pumped from the slakers to two lime slurry storage tanks from which it is pumped to the neutralization streams of the treatment circuits as a 20% solids slurry.

Barium chloride, received in bags, is manually made-up as a 5% solution in a small mix tank and transferred to a holding tank for metering to the raffinate and minewater treatment circuits.

Ferric sulphate is received in bulk as ferri-floc and made up as a 7% solution for feeding to the minewater treatment circuit.

Flocculant is made up with water using an aspirator and diluted to 0.05% in a flocculant mixing tank. This solution is transferred to a holding tank for feeding to various dewatering and clarification points in the waste treatment circuits.

## 6.4.5 (Cont'd)

Nitric acid is received by tank truck as 65-70% $HNO_3$ and unloaded into a stainless steel holding tank. An in-line mixer is used to dilute the acid to 4% for use in acid treating the sand filters in the raffinate and minewater treatment circuits.

Low pressure air is supplied to the drum filters by a single 500 $m^3/h$ air compressor. Air for the neutralization pachucas, the sand filters and miscellaneous requirements is supplied by a single 1,700 $m^3/h$ air compressor. Standby air and instrument air is assumed available from the main process air supply.

Steam supply for the ammonium sulphate crystallization plant and for use in controlling temperatures of the tailings slurry is assumed available from waste heat recovery from other areas of the operating facility.

Propane for the ammonium sulphate dryer is supplied from the main propane tank farm on site which also supplies propane for heating requirements.

Power supply for the property is diesel generated due to the remoteness of the operation.

Process water requirements for the waste treatment circuits are minimal and water supply for reagent make-up is assumed available from the main process plant water supply.

## 6.5 TAILINGS CONTAINMENT

The Model C tailings containment basin is constructed on gently rolling topography similar to the southern portion of the Athabasca basin of north-central Saskatchewan. The surface landscape is composed of glacially-deposited sand and sand-till features combined with scattered lakes and muskeg depressions. Bedrock is assumed to be well below any required excavations. Vegetation consists mainly of swamp spruce in low-lying areas.

Construction of the tailings basin is along a shallow hillside in order to utilize excavated materials for embankment construction. The method of tailings deposition is the sub-aerial technique incorporating an extensive underdrainage system beneath the entire base of the facility. The bottom of the basin is covered by a graded underseal with a filter blanket and perforated drainage system overlying this seal. The underdrainage system serves to intercept all seepage from the base of the tailings and, by removing any build-up of head within the filter blanket, significantly reduces seepage across the seal. The underseal beneath the facility, and seals within the main embankment, are constructed by in-situ modification of the till material with bentonite, moisture conditioning and compaction. All dyke construction material is assumed to be available adjacent to the tailings facility. A general arrangement and cross sections of the tailings facility are shown on Drawing Nos. 100-35-5 and 100-35-6.

The tailings are pumped to the tailings storage facility at a temperature of 30°C in either of two 100 mm diameter steel tailings lines. Tailings pumps are the positive displacement glandless diaphram type with one operating and one standby for emergencies.

**6.5 (Cont'd)**

Tailings are deposited in thin layers from spray bars for at least 8 months of the year along the upstream end of the facility and flow towards the main embankment to produce an exposed tailings beach with a slope of approximately 1 in 100. During winter months tailings are open-end discharged in selected areas of the tailings basin to form a thicker winter layer that will be left exposed in the following summer until it has thawed, drained and consolidated. Precipitation and supernatant water flow down the beach towards the main embankment.

The main embankment incorporates a pervious upstream zone which continuously decants precipitation and supernatant water into a system of collection pipes leading through the embankment. Continuous decanting through the pervious zone eliminates surface ponding and contributes to the drained condition of the tailings.

Surface water and seepage from the underdrainage system are collected in a central sump and are pumped either to a holding tank in the mill for feeding to the raffinate treatment circuit or to the upstream embankment of the pond for tailings wetting and additional evaporation availability.

The water return line from the central sump to the mill is enclosed in a metal covered concrete utilidor together with the tailings lines and a water recycle line. The operating tailings line provides sufficient heat to prevent freezing of the water lines during winter months. It is possible to recycle back to the tailings pond excess water from the mill, or contaminated water unsuitable for release to the environment, by the use of the recycle water line.

**6.5 (Cont'd)**

Additional features constructed in the tailings management area include the following:

- access road from the mill to the tailings pond including adjacent bench for pipe utilidor
- three phase, above-grade power supply to the seepage and runoff sump at the tailings area
- emergency sump at the mid-point between the mill and tailings area including water connection for tailings line flush.

The principal design objectives of this type of facility are to achieve a system whereby seepage discharge to the environment is reduced during operation and to ensure that at decommissioning the tailings will be drained and consolidated. Decommissioning of this basin is expected to be relatively inexpensive and construction of a surface seal would eliminate any long-term seepage.

## 6.6 MONITORING REQUIREMENTS

The environmental monitoring requirements for Model C have been detailed according to Saskatchewan Environment and AECB regulations. The sample locations, the constituent analysis, the frequency of sampling and the types of samples required are listed in Table C-6 in Appendix C.

The sample stations for liquid samples in Model C include:

- treated effluent (1)
- seepage and runoff from the tailings basin (1)
- minewater (1)
- tailings area piezometers (15)
- pond area piezometers (5)
- ore stockpile seepage collection (1)
- dewatering well (1)
- downstream stations (14)

Other sample requirements include 7 stations for biota sampling, 4 stations for sediment sampling, 5 stations for soil sampling and 2 stations for plant sampling.

A summary of analytical requirements (Tables C-7, C-8 and C-9) indicates that a total of 11,325 determinations are required on an annual basis. A round figure of 10,000 determinations per year has therefore been assumed for Model C monitoring requirements. As in Models A and B the site assay office would perform all environmental analysis.

## 6.7 CAPITAL COSTS

### 6.7.1 Summary

The capital cost summary presented in Table 6.1 provides the estimated costs for the Model C waste management system for a 466 t/day ore processing facility. These estimated costs total $47,665,000 in first quarter 1984 Canadian dollars and provide for all costs of the waste management aspects of both minewater and mill tailings treatment.

The estimate includes:

- building structures
- minewater treatment circuit
- mill effluent treatment circuit
- reagents and utilities
- tailings lines
- tailings storage basin
- monitoring
- indirect costs of construction
- engineering, procurement, and construction management
- contingency allowance.

The detailed capital cost estimate for Model C is provided under separate cover.

A breakdown of the capital costs for each major area is presented in Table 6.2. These costs are based on the total capital costs with indirect capital costs added to the direct capital costs of each area on a proportioned basis. Unit capital costs relative to the annual uranium production and annual tonnage throughput of the facility are also presented by area.

# TABLE 6.1

## MODEL C
## CAPITAL COST ESTIMATE

| | |
|---|---|
| **Direct Costs** | |
| Minewater Treatment | $ 3,590,000 |
| Mill Effluent Treatment | 17,091,000 |
| Tailings Line | 1,719,000 |
| Tailings Storage Basin | 11,938,000 |
| Monitoring | 1,213,000 |
| Total Direct Costs | $ 35,501,000 |
| **Indirect Costs** | |
| Construction Indirects (8%) | $ 2,840,000 |
| Engineering, Procurement and Construction Management (12%) | 4,260,000 |
| Capital Spares | 731,000 |
| Subtotal | 43,332,000 |
| Contingency (10%) | 4,333,000 |
| TOTAL MODEL C CAPITAL COSTS | $ 47,665,000 |

## TABLE 6.2

## MODEL C
## CAPITAL COSTS BY AREA

| Area | Dollars | \$/annual tonne | \$/annual kg U | \$/annual lb $U_3O_8$ |
|---|---|---|---|---|
| Minewater Treatment | 4,820,100 | 31.08 | 2.51 | 0.96 |
| Mill Effluent Treatment | 22,879,900 | 147.54 | 11.90 | 4.58 |
| Tailings Line | 2,308,000 | 14.88 | 1.20 | 0.46 |
| Tailings Storage Basin | 16,028,400 | 103.36 | 8.33 | 3.21 |
| Monitoring | 1,628,600 | 10.50 | 0.85 | 0.33 |
| TOTAL | 47,665,000 | 307.36 | 24.79 | 9.54 |

### 6.7.1 (Cont'd)

On this basis, the total unit capital costs for Model C are:

- $307.36 per annual tonne of ore processed
- $24.79 per annual kilogram of U produced
- $9.54 per annual pound of $U_3O_8$ produced.

These cost estimates do not include any provision for pre-production construction interest as the total value of this interest is dependent on the method of financing and the equity or credit position of the specific investor. These pre-production interest costs can have a significant impact on the total costs of any project and this should be kept in mind when reviewing the cost estimates given in this report.

### 6.7.2 Basis of Estimate

The capital cost estimate is based on the following information:

- flowsheets and conceptual layouts prepared by Kilborn and Melis Consulting Engineers Ltd.
- process and material specifications as supplied by Melis Consulting Engineers Ltd.
- estimated construction material quantities as derived by Kilborn and Knight and Piesold Ltd.
- budget quotations on major equipment
- pricing data from Kilborn's files and records.
- tailings basin design and pricing data from Knight and Piesold Ltd.

**6.7.2 (Cont'd)**

For purposes of this study, it has been assumed that:

- all construction services would be tendered by fixed price contract. The average labour rate is $50.00 per hour based on a 60-hour work week. This labour rate includes provision for direct wages, payroll burdens and benefits, small tool and consumables, site supervision, subsistence, travel time and expenses, and the contractor's overhead and fee
- construction labour force would be transported to site and work a three-week in/one-week out schedule and reside in a construction camp while on site
- all costs are based on first quarter 1984 Canadian dollars
- the cumulative price factor, applied to base prices for process equipment is nineteen percent. This includes:

  - provincial sales tax - 5 %
  - freight allowance - 8 %
  - miscellaneous minor materials allowance - 5 %.

No allowances have been made in the capital estimate for:

- escalation beyond the first quarter 1984
- preproduction construction interest
- working capital
- production equipment or materials
- purchase of land or right-of-way easements
- start-up assistance
- cost of pre-appropriation studies, development work or environmental studies and hearings
- owners' costs such as an owners' representative

### 6.7.2 (Cont'd)

- general property facilities such as housing and administration offices for operations personnel
- pumping minewater from the mine to the treatment facility
- decommissioning.

### 6.7.3 Methodology

Equipment prices were obtained on an individual basis from budget quotations or historical data. Mechanical installation costs, including field material, were factored from the equipment costs. In-plant piping, electrical and instrumentation costs were factored from the total installed mechanical costs. Building structure costs for each area were based on volumes and volume unit prices derived from historical data. Building costs include the cost of the building for minewater treatment, mill tailings and effluent treatment and the ammonium sulphate crystallization plant.

The monitoring cost include proportional costs of a laboratory building and laboratory equipment costs for a site analytical laboratory. It is assumed that thirty percent of analytical efforts for a uranium operation are dedicated to environmental requirements and this portion of the capital cost of the laboratory building and equipment is included in this area. It also includes the cost of the monitoring ponds, recycle water pumps, the seepage and runoff pumping circuit and piezometers and sampling wells for the tailings area.

The allowance for construction indirects has been estimated as eight percent of the total direct costs. It includes:

- temporary site services
- permits, insurance and bonds

## 6.7.3 (Cont'd)

- winter works
- rental of mobile equipment
- on site construction supervision facilities
- pre-construction geotechnical and hydrogeotechnical investigations.

The engineering, procurement and construction management function has been estimated as twelve percent of the direct costs. As well as design engineering, this includes provision for project management, cost control, scheduling, contract administration and inspection services.

An allowance of five percent of home office costs has been included for capital spares inventory. The contingency allowance provides for additions and unidentified items within the stated scope of work but it is not intended to cover major changes in scope.

## 6.8 OPERATING COSTS

### 6.8.1 Summary

The total waste management operating cost for Model C, as estimated from the design basis given in Section 6.3, is \$8,579,667 per year in first quarter 1984 dollars. This total, which includes a five percent contingency allowance, is equivalent to a cost of \$55.32/tonne of ore processed or \$4.46/kg U (\$1.72/lb $U_3O_8$) based on an annual uranium production of 1,923,000 kg U (5,000,000 lbs $U_3O_8$).

The Model C operating costs are summarized in Table 6.3 and an estimated breakdown of the costs for each area of the waste management system is provided in Table 6.4. Cost details are given in Tables 6.5, 6.6 and 6.7.

The operating cost breakdown given in Table 6.4 is a proportioned estimate of the total cost allocated to the following areas:

- minewater treatment
- mill effluent treatment
- tailings line
- tailings basin
- monitoring.

The cost of reagents and supplies has been proportioned according to usage for each area.

The minewater treatment operating costs include all associated costs of the separate minewater treatment circuit. The mill effluent treatment operating costs include the cost of treating all mill

## TABLE 6.3

## MODEL C
## OPERATING COST ESTIMATE

| Item | Annual $ | $/tonne | $/kg U | $/lb $U_3O_8$ |
|---|---|---|---|---|
| Manpower | 1,353,460 | 8.72 | 0.71 | 0.28 |
| Waste Treatment | 6,557,651 | 42.29 | 3.41 | 1.31 |
| Roads & Yards | 40,000 | 0.26 | 0.02 | 0.01 |
| Accommodation | 155,000 | 1.00 | 0.08 | 0.03 |
| Transportation | 65,000 | 0.42 | 0.03 | 0.01 |
| Contingency (5%) | 408,556 | 2.63 | 0.21 | 0.08 |
| TOTAL | 8,579,667 | 55.32 | 4.46 | 1.72 |

## TABLE 6.4

## MODEL C
## ANNUAL OPERATING COSTS BY AREA ($)

| Item | Minewater Treatment | Mill Effluent Treatment | Tailings Line | Tailings Basin | Monitoring | Total |
|---|---|---|---|---|---|---|
| Manpower | 295,562 | 541,862 | 49,260 | 98,520 | 368,256 | 1,353,460 |
| Reagents & Supplies | 835,233 | 3,744,501 | 10,000 | 20,000 | 20,000 | 4,629,734 |
| Power | 243,651 | 1,216,711 | - | - | 80,000 | 1,540,362 |
| Heating | 78,369 | 307,686 | - | - | 1,500 | 387,555 |
| Roads & Yards | 5,000 | - | 10,000 | 20,000 | 5,000 | 40,000 |
| Accomodation & Transportation | 47,960 | 88,000 | 7,920 | 16,060 | 60,060 | 220,000 |
| SUB-TOTAL | 1,505,775 | 5,898,760 | 77,180 | 154,580 | 534,816 | 8,171,111 |
| Contingency (5%) | 75,289 | 294,938 | 3,859 | 7,729 | 26,741 | 408,556 |
| TOTAL | 1,581,064 | 6,193,698 | 81,039 | 162,309 | 561,557 | 8,579,667 |
| $/tonne | 10.19 | 39.94 | 0.52 | 1.05 | 3.62 | 55.32 |
| $/kg U | 0.82 | 3.22 | 0.04 | 0.09 | 0.29 | 4.46 |
| $/lb $U_3O_8$ | 0.32 | 1.24 | 0.02 | 0.03 | 0.11 | 1.72 |

## TABLE 6.5

## MODEL C
## ANNUAL MANPOWER COSTS

| Employee | Requirement | Number | Rate | Annual $ |
|---|---|---|---|---|
| Supervision: | | | | |
| - Operational | 0.25 man/shift | 1.0 | $3,675/month | 44,100 |
| - Maintenance | 0.25 man/day | 1.0 | $3,675/month | 44,100 |
| - Environmental | 0.5 man/day | 1.0 | $3,675/month | 44,100 |
| - Process Engineer | 1.0 man/day | 2.0 | $3,415/month | 81,960 |
| - Analytical | 0.4 man/day | 0.8 | $3,415/month | 32,785 |
| Environmental Sampler | 1.0 man/day | 2.0 | $2,625/month | 63,000 |
| Analytical Technician | 2.0 men/day | 4.0 | $2,890/month | 138,720 |
| Control Room Operator | 1.0 man/shift | 4.0 | $16.80/hour | 129,360 |
| Floor Operator | 1.5 men/shift | 6.0 | $14.70/hour | 169,785 |
| Tails & Pond Area Operator | 0.5 man/day | 2.0 | $14.70/hour | 56,595 |
| Reagent Operator | 1.0 man/shift | 2.0 | $13.50/hour | 51,975 |
| Crystallizer Operator | 1.0 man/shift | 4.0 | $14.70/hour | 113,190 |
| Waste Treatment Area Maintenance | 45 mandays/month | 3.0 | $16.80/hour | 99,790 |
| Tails Area Maintenance | 6 mandays/month | 0.8 | $16.80/hour | 13,305 |
| SUB-TOTAL | | 33.6 | | 1,082,765 |
| Payroll Burden (25%) | | | | 270,695 |
| TOTAL | | | | 1,353,460 |

## TABLE 6.6

## MODEL C
## WASTE TREATMENT COSTS

| Item | Annual $ | $/tonne | $/kg U | $/lb $U_3O_8$ |
|---|---|---|---|---|
| Reagents & Supplies | 4,629,734 | 29.86 | 2.41 | 0.92 |
| Power | 1,540,362 | 9.93 | 0.80 | 0.31 |
| Heating | 387,555 | 2.50 | 0.20 | 0.08 |
| TOTAL | 6,557,651 | 42,29 | 3.41 | 1.31 |

## TABLE 6.7

## MODEL C
## REAGENTS AND SUPPLIES

| Item | Unit Cost ($/t) | Consumption kg/t* | Consumption t/a | Annual $ |
|---|---|---|---|---|
| Lime (95% CaO) | 150 | 155.3 | 24,083 | 3,612,450 |
| Barium Chloride | 411 | 0.99 | 152.9 | 62,842 |
| Flocculant | 4,500 | 0.04 | 6.646 | 29,907 |
| Ferri-Floc | 385 | 10.7 | 1,653 | 636,405 |
| Nitric Acid | 576 | 0.5 | 80 | 46,080 |
| Propane | 0.21/L | | 105 $m^3$/a | 22,050 |
| Laboratory Supplies | | | | 20,000 |
| Maintenance Supplies | | | | 150,000 |
| Operating Supplies | | | | 50,000 |
| TOTAL | | | | 4,629,734 |

* Reagent consumption based on 155,078 tonnes of mill feed per year.

### 6.8.1 (Cont'd)

waste solutions as well as the cost of operating the ammonium sulphate crystallization plant. The tailings line and tailings basin operating costs are the manpower and miscellaneous costs associated with these areas. The monitoring costs include environmental sampling and analysis as well as the costs associated with the monitoring ponds and water recycling.

### 6.8.2 Manpower

The manpower costs detailed in Table 6.5 reflect the costs of operating all waste treatment related circuits for Model C as well as the tailings area and efforts expended for meeting monitoring requirements. The annual total of $1,353,460 includes a twenty-five percent allowance for payroll burden. The salary rates are applicable to a northern Saskatchewan operation and are the same rates as used in Model B.

As in Model B, the total number of employees is double the manpower allowance to have full year-round coverage with a 7-day in, 7-day out work schedule. A total of 33.6 employees has been estimated for Model C waste management.

The allowance for Model C supervision is greater than the allowance for Model A or B since the Model C waste management system is more extensive than the systems in the other two models. Operational and maintenance supervision is estimated at twenty-five percent of a supervisor's time for waste management related areas. Environmental supervision is provided by allowing 0.5 man per day with analytical supervision being covered by 0.4 man per day. One process engineer

### 6.8.2 (Cont'd)

on dayshift is included for process and troubleshooting work for the waste management circuits.

One environmental sampler spends full time on sampling and field measurements to meet all the monitoring requirements as detailed in Appendix C. The analytical workload is covered by two technicians on dayshift.

A total of 4 men per shift has been included for operating the minewater treatment circuit, the raffinate treatment circuit, the tailings neutralization circuit, the ammonium sulphate crystallization circuit, the reagent make-up facilities and the tailings area. Maintenance coverage for all waste management areas include 51 mandays per month. The work schedule for hourly employees is 2 shifts per day, 11 hours per shift, 350 days per year.

### 6.8.3. Reagents and Supplies

The annual cost of reagents and supplies ($4,629,734) is based on the consumption figures used for Model C and the delivered price of reagents at the Model C site. The annual reagent requirements are based on an annual tonnage of 155,078 tonnes.

The lime costs include the requirements for raffinate treatment, tailings neutralization and minewater treatment. Barium chloride, flocculant, ferri-floc and nitric acid meet the requirements for both the raffinate treatment circuit and the minewater treatment circuit.

### 6.8.3 (Cont'd)

The propane cost only includes the propane requirements for the ammonium sulphate dryer. Lump sum estimates have been included for laboratory, maintenance and operating supplies for the waste management areas of the Model C facility. The consumption of sulphuric acid for pH adjustment has been assumed as negligible and is not included in the costs.

### 6.8.4 Power

The total power requirements for the waste management section of the Model C facility is estimated at 1,417 kW (1,900 HP). The annual power cost is $1,540,362 based on an 85% power efficiency factor and a 350 day per year operation with a $0.11/kWh diesel-generated power unit cost.

### 6.8.5 Heating

The annual heating costs of $387,555 include heating for the waste treatment building and a proportion of the site laboratory building. Annual propane requirements are estimated at 1,845,500 L at a cost of $0.21/L. This includes additional heating to allow for sufficient air changes in the waste treatment building for potential radon gas control requirements.

### 6.8.6 Miscellaneous

A lump sum allowance of $40,000 has been included for the cost of maintaining roads and yards associated with the waste management section.

**6.8.6 (Cont'd)**

Accommodation costs of $155,000 per year include the cost of room and board for waste management personnel based on 16.8 beds at a rate of $25.00 per bed per day. Employee transportation costs of $65,000 per year have been estimated for 33.6 employees with 23 round trips per year per employee at a rate of $80 per round trip.

The cost of shipping ammonium sulphate product to market has not been included as an operating cost based on the assumption that revenue from the sale of this product would offset any transportation cost. No net revenue has been included for offsetting operating costs of the ammonium sulphate circuit.

## 6.9 CASH FLOW AND NET PRESENT VALUE

A net cash flow for the Model C waste management system is presented in Table 6.8. This cash flow is based on the construction schedule indicated in the terms of reference for this study. The key dates are:

- Engineering Start: January 1980
- Construction Start: June 1981
- Plant Startup: August 1983
- Plant decommissioning: 2004

It has also been assumed that no significant production would occur during the period August 1983 to December 1983 and there would therefore be no significant operating costs during this period.

Normally a total cash flow is the summation of revenue, operating costs, interest expenses, freight and transportation, fixed capital, working capital, sustaining capital, federal taxation, provincial taxation and selling expenses.

Since this study presents a net cash flow for waste management only, the net cash flow is defined as the total project cash flow including the waste management costs minus the total project cash flow excluding the waste management costs. Therefore, in the net cash flow calculation, all costs that are not affected by waste management, such as revenue, freight, transportation and selling expenses, can be excluded from consideration.

## 6.9 (Cont'd)

Since the study criteria also excluded taxation considerations federal and provincial taxes have been excluded in the calculations. In addition, since interest expenses are dependent on financing and are investor-specific they are also excluded in this analysis.

Consequently, the net cash flow equation used for calculating the incremental waste management costs is:

Cash Flow = operating costs + fixed capital + working capital + sustaining capital.

For Model C sustaining capital is negligible and it does not enter into the calculation.

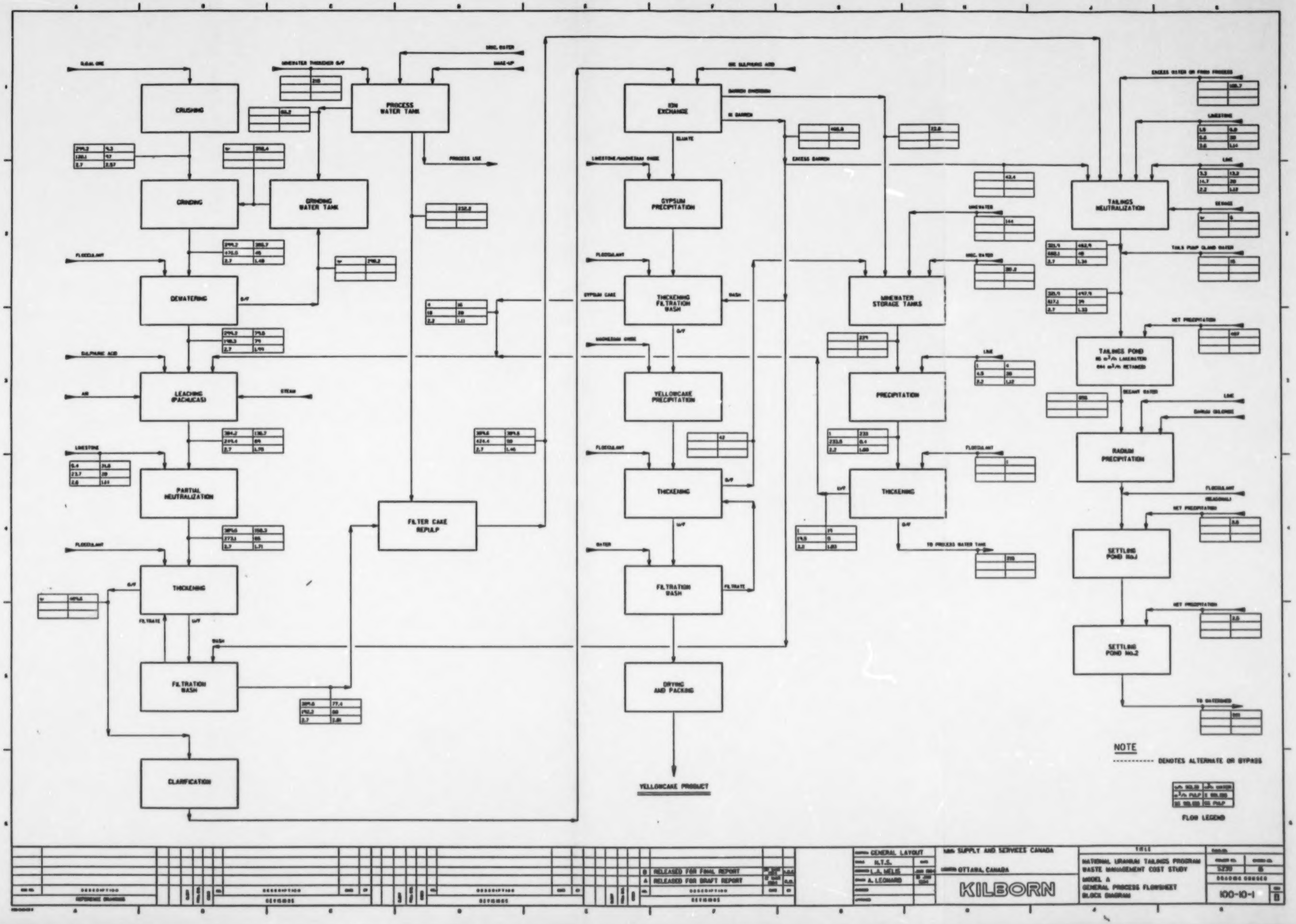
A net present value of minus $81,090,000 has been determined for Model C based on the cost elements of the net cash flow given in Table 6.8. This net present value is based on a discount rate of 12.5% and gives the net present value in 1984 constant dollars at the date in time when the initial capital is committed (i.e. 1980). Since this calculation is on a pre-tax basis, no consideration is given for capital cost allowance or depreciation.

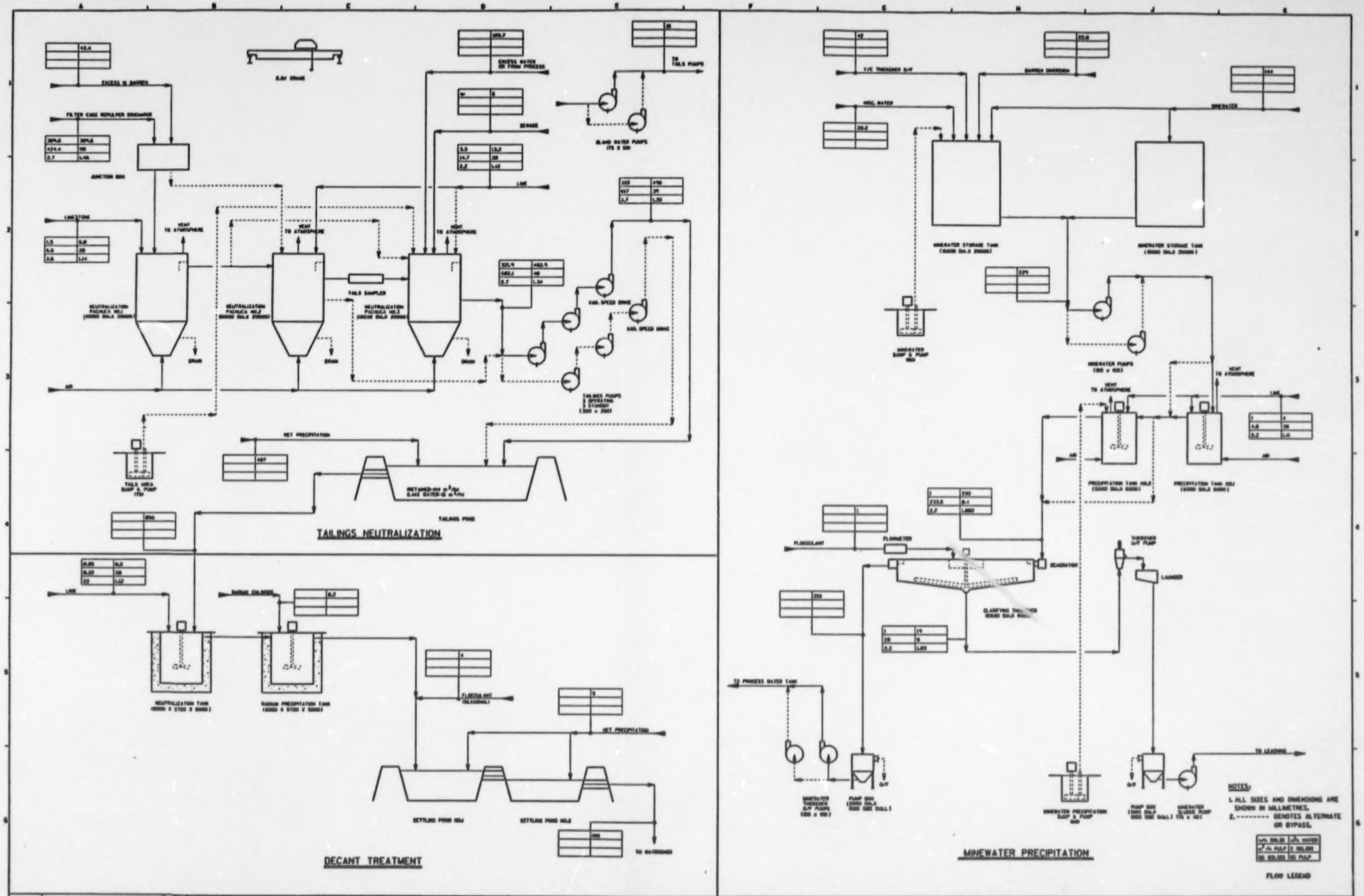
## TABLE 6.8

### MODEL C
### CASH FLOW
(CONSTANT 1984 $ x 1,000,000)

| | 1980 | 1981 | 1982 | 1983 | 1984 | 1985 | 1986 | 1987 | 1988 | 1989 | 1990 | 1991 | 1992 | 1993 | 1994 | 1995 | 1996 | 1997 | 1998 | 1999 | 2000 | 2001 | 2002 | 2003 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 |
| Operating Cost | | | | | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 |
| Capital Cost: | | | | | | | | | | | | | | | | | | | | | | | | |
| - initial | 1.0 | 9.6 | 19.2 | 17.9 | | | | | | | | | | | | | | | | | | | | |
| - sustaining | | | | | | | | | | | | | | | | | | | | | | | | |
| Working Capital | | | | 0.5 | | | | | | | | | | | | | | | | | | | | |
| Total Costs | 1.0 | 9.6 | 19.2 | 18.4 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 | 8.6 |
| Present Value Factor | 1.0 | 0.89 | 0.79 | 0.70 | 0.62 | 0.55 | 0.49 | 0.44 | 0.39 | 0.35 | 0.31 | 0.27 | 0.24 | 0.22 | 0.19 | 0.17 | 0.15 | 0.14 | 0.12 | 0.11 | 0.09 | 0.08 | 0.07 | 0.07 |
| Present Value | 1.0 | 8.54 | 15.17 | 12.88 | 5.33 | 4.73 | 4.21 | 3.78 | 3.35 | 3.01 | 2.67 | 2.32 | 2.06 | 1.89 | 1.63 | 1.46 | 1.25 | 1.20 | 1.03 | 0.95 | 0.77 | 0.69 | 0.60 | 0.60 |
| Net Present Value | 1.0 | 9.54 | 24.71 | 37.59 | 42.89 | 47.62 | 51.83 | 55.61 | 58.96 | 61.97 | 64.64 | 66.96 | 69.02 | 70.91 | 72.54 | 74.00 | 75.25 | 76.45 | 77.48 | 78.43 | 79.20 | 79.89 | 80.49 | 81.09 |

# DRAWINGS


R.O.M. ORE
CRUSHING
MINEWATER THICKENER O/F
210
PROCESS WATER TANK
MISC. WATER
MAKE-UP
60.2
PROCESS USE
299.2 | 9.3
120.1 | 97
2.7 | 2.57
W | 258.4
GRINDING
GRINDING WATER TANK
232.8
299.2 | 385.7
470.0 | 45
2.7 | 1.48
FLOCCULANT
W | 298.2
DEWATERING
O/F
4 | 16
18 | 20
2.2 | 1.11
299.2 | 79.0
198.3 | 79
2.7 | 1.99
SULPHURIC ACID
LEACHING (PACHUCAS)
AIR
STEAM
384.2 | 130.7
249.4 | 89
2.7 | 1.70
309.6 | 309.6
424.4 | 50
2.7 | 1.46
LIMESTONE
5.4 | 21.6
23.7 | 20
2.8 | 1.11
PARTIAL NEUTRALIZATION
309.6 | 158.3
273.1 | 65
2.7 | 1.71
FLOCCULANT
THICKENING
O/F
W | 109.6
FILTRATE
U/F
WASH
FILTRATION WASH
309.6 | 77.4
192.2 | 80
2.7 | 2.01
FILTER CAKE REPULP
CLARIFICATION
ION EXCHANGE
IX SULPHURIC ACID
BARREN OVERFLOW
IX BARREN
ELUATE
460.8
22.8
EXCESS BARREN
LIMESTONE/MAGNESIUM OXIDE
GYPSUM PRECIPITATION
FLOCCULANT
GYPSUM CAKE
THICKENING FILTRATION WASH
WASH
O/F
MAGNESIUM OXIDE
YELLOWCAKE PRECIPITATION
42
FLOCCULANT
THICKENING
O/F
U/F
WATER
FILTRATION WASH
FILTRATE
DRYING AND PACKING
YELLOWCAKE PRODUCT
42.4
MINEWATER
144
MISC. WATER
20.2
MINEWATER STORAGE TANKS
229
LIME
1 | 4
4.5 | 20
2.2 | 1.12
PRECIPITATION
1 | 233
233.8 | 0.4
2.2 | 1.00
FLOCCULANT
1
U/F
THICKENING
O/F
1 | 21
19.0 | 5
2.2 | 1.03
TO PROCESS WATER TANK
210
EXCESS WATER OR FROM PROCESS
105.7
LIMESTONE
1.5 | 6.0
6.0 | 20
2.6 | 1.11
LIME
3.3 | 13.2
14.7 | 20
2.2 | 1.12
TAILINGS NEUTRALIZATION
SEWAGE
W | 5
321.9 | 482.9
660.1 | 40
2.7 | 1.30
TAILS PUMP GLAND WATER
15
321.9 | 497.9
617.1 | 39
2.7 | 1.33
NET PRECIPITATION
487
TAILINGS POND
85 m³/h LAKEWATER
814 m³/h RETAINED
DECANT WATER
816
LIME
BARIUM CHLORIDE
RADIUM PRECIPITATION
FLOCCULANT (SEASONAL)
NET PRECIPITATION
3.8
SETTLING POND No.1
NET PRECIPITATION
2.0
SETTLING POND No.2
TO WATERSHED
901
NOTE
------------ DENOTES ALTERNATE OR BYPASS
t/h SOLID | t/h WATER
m³/h PULP | % SOLIDS
SG SOLIDS | SG PULP
FLOW LEGEND
B | RELEASED FOR FINAL REPORT
A | RELEASED FOR DRAFT REPORT
REFERENCE DRAWINGS
REVISIONS
GENERAL LAYOUT
N.T.S.
L.A. MELIS
A. LEONARD
SUPPLY AND SERVICES CANADA
OTTAWA, CANADA
KILBORN
TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL A
GENERAL PROCESS FLOWSHEET
BLOCK DIAGRAM
5230
B
100-10-1
B



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| A | RELEASED FOR DRAFT REPORT | | | |
| NO. | DESCRIPTION | DATE | BY | |

REVISIONS

REFERENCE DRAWINGS

GENERAL LAYOUT
N.T.S.
L.A. MELIS
R.BAIRD

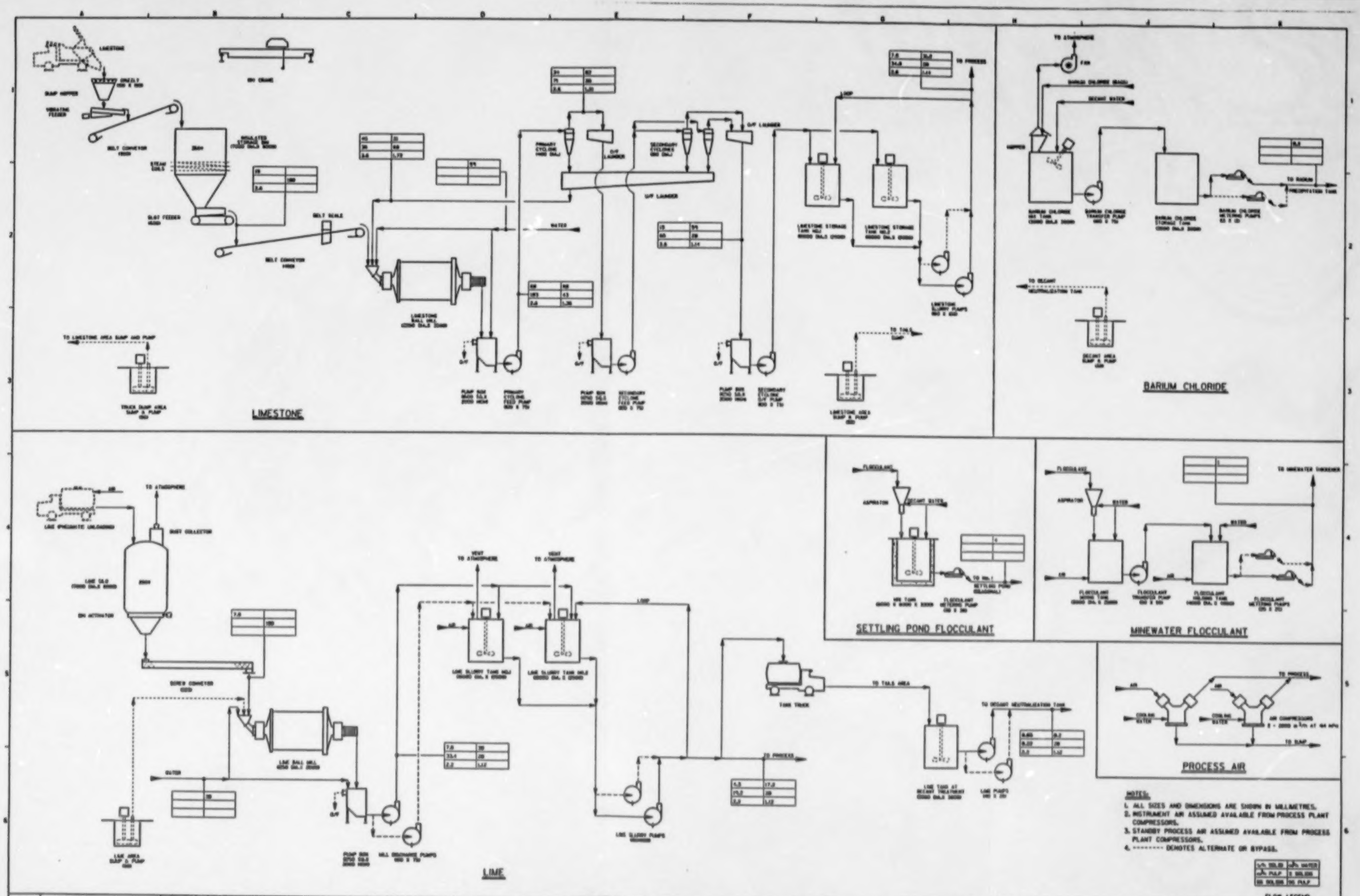
SUPPLY AND SERVICES CANADA.
OTTAWA, CANADA

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL A
WASTE MANAGEMENT FLOWSHEET

5239 | 15

DRAWING NUMBER
100-10-2 | B



NOTES:

1. ALL SIZES AND DIMENSIONS ARE SHOWN IN MILLIMETRES.
2. INSTRUMENT AIR ASSUMED AVAILABLE FROM PROCESS PLANT COMPRESSORS.
3. STANDBY PROCESS AIR ASSUMED AVAILABLE FROM PROCESS PLANT COMPRESSORS.
4. --------- DENOTES ALTERNATE OR BYPASS.

| | |
|---|---|
| t/h SOLIDS | m³/h WATER |
| m³/h PULP | % SOLIDS |
| SG SOLIDS | SG PULP |

FLOW LEGEND

| REV | DESCRIPTION | DATE | BY |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | | D.A. |
| A | RELEASED FOR DRAFT REPORT | | D.A. |

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GENERAL LAYOUT

M.T.S.

L.A. MELIS

R. BAIRD

CLIENT: SUPPLY AND SERVICES CANADA

LOCATION: OTTAWA, CANADA

KILBORN

TITLE

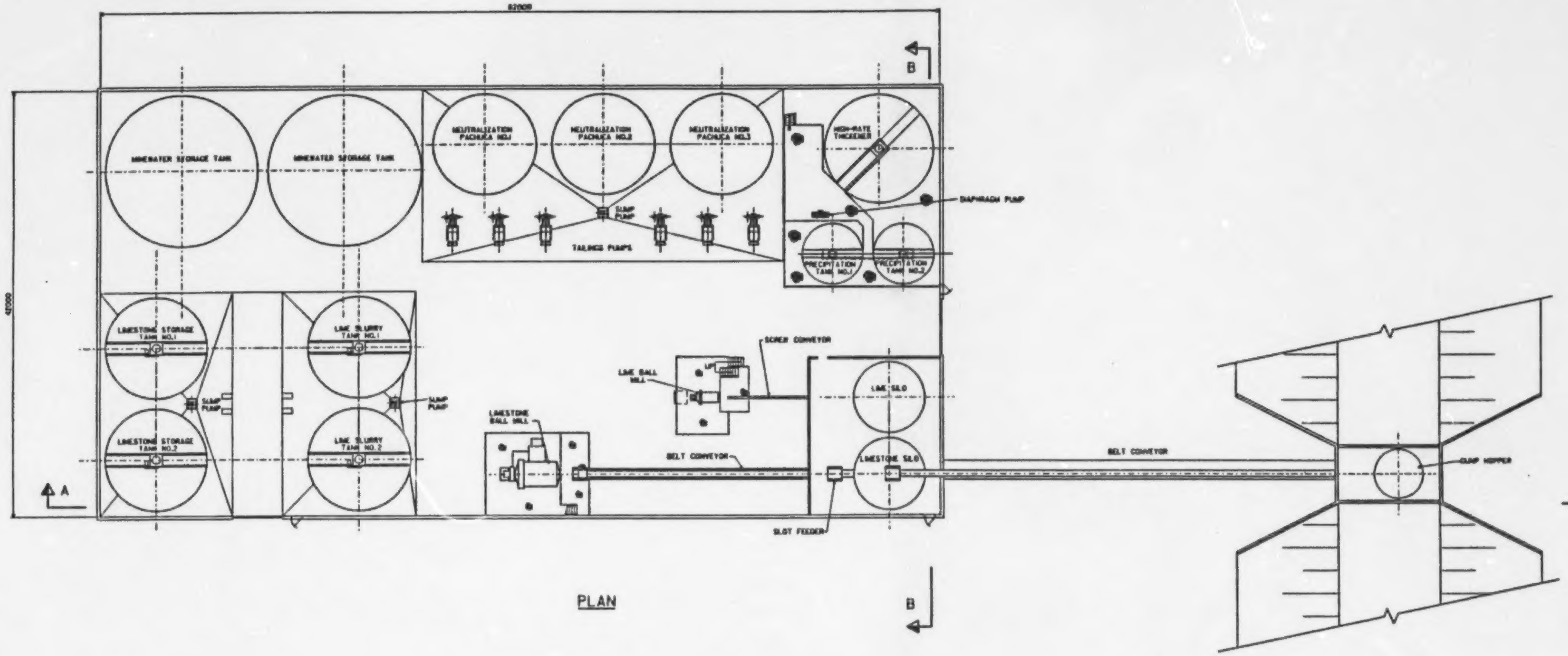
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL A
REAGENT FLOWSHEET

5239 | 6

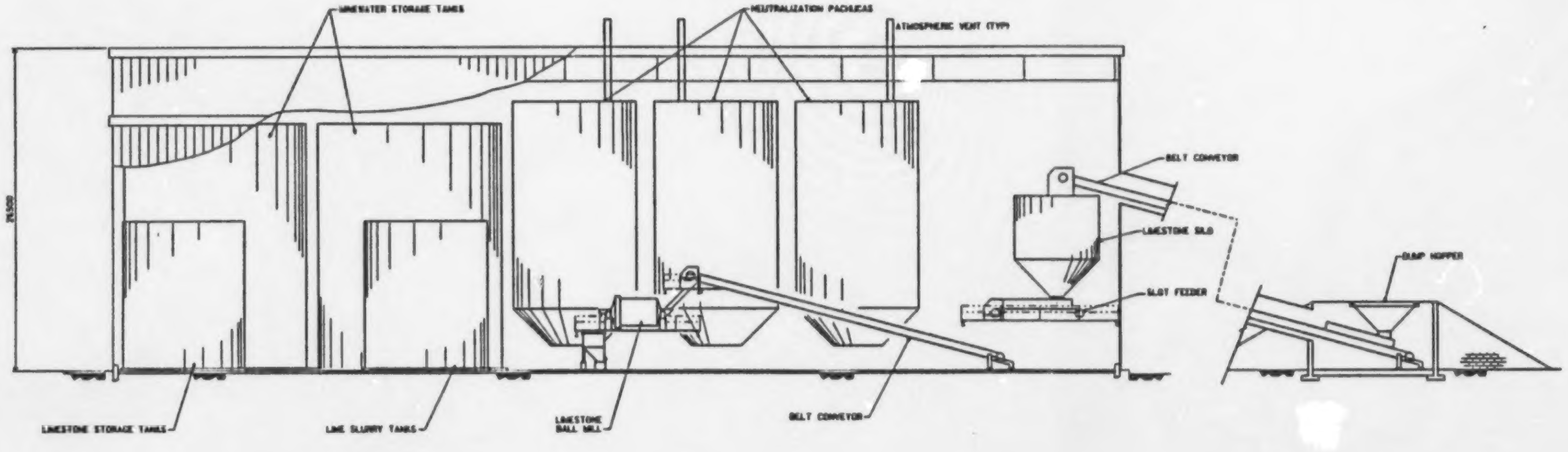
DRAWING NUMBER

100-10-3

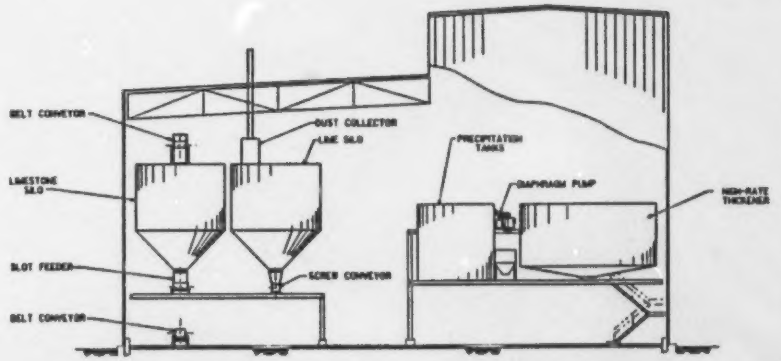
B



PLAN



SECTION A-A



SECTION B-B

NOTE
1) ALL DIMENSION ARE IN MILLIMETRES

0 2000 10000 mm
SCALE 1:200

| NO. | DESCRIPTION | DATE | BY |
|---|---|---|---|
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| A | RELEASED FOR DRAFT REPORT | [illegible] | A.L. |

REFERENCE DRAWINGS
REVISIONS

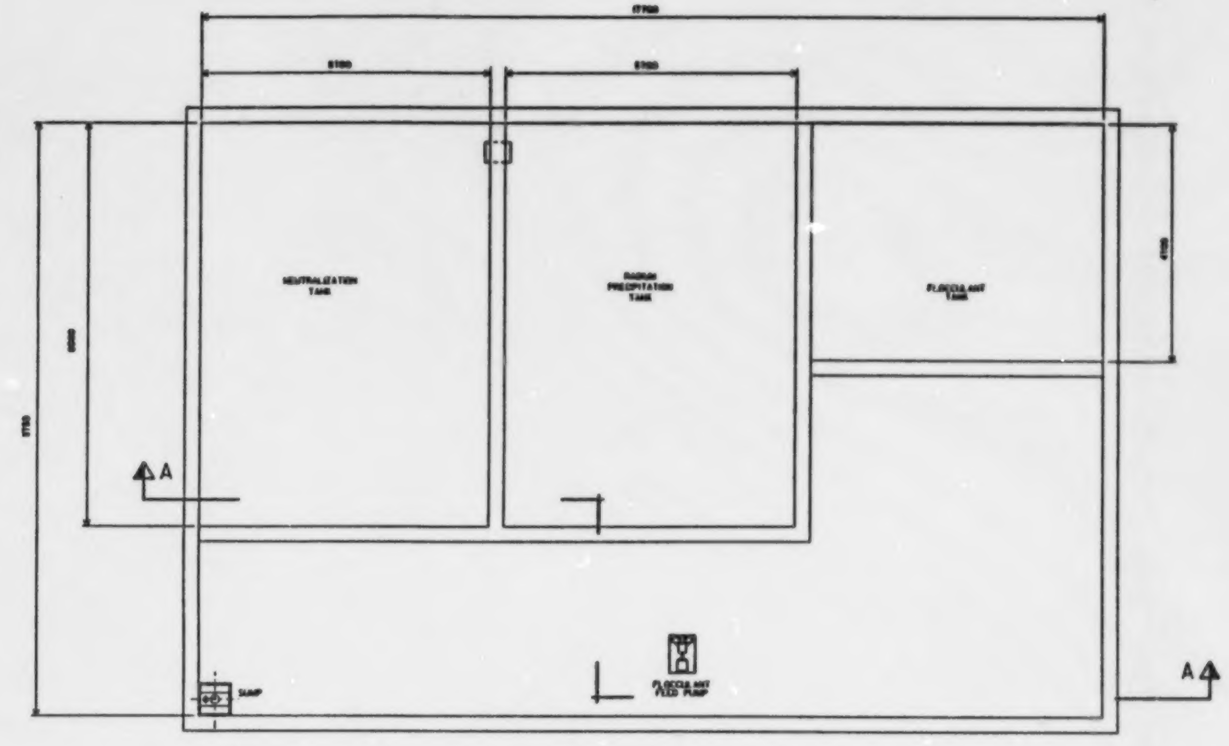
GENERAL LAYOUT
SCALE 1:200
DESIGNED T. WALBAUM 30 JAN 1984
DRAWN A. LEONARD 31 JAN 1984
CHECKED
APPROVED

CLIENT: SUPPLY AND SERVICES CANADA
LOCATION: OTTAWA, CANADA

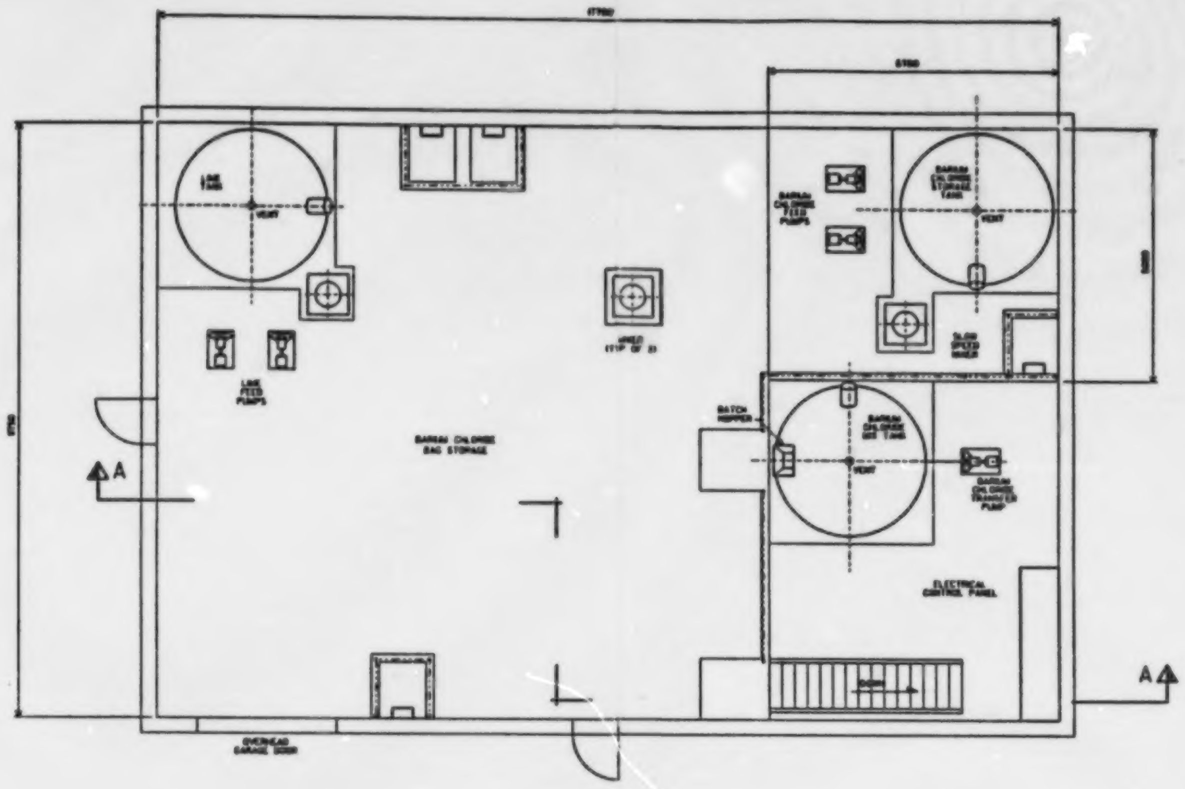
KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL A
WASTE TREATMENT BUILDING
PLAN AND SECTIONS

PROJECT NO. S239
DIVISION NO. 15
DRAWING NUMBER 100-10-4
REV. B



PLAN (LOWER)



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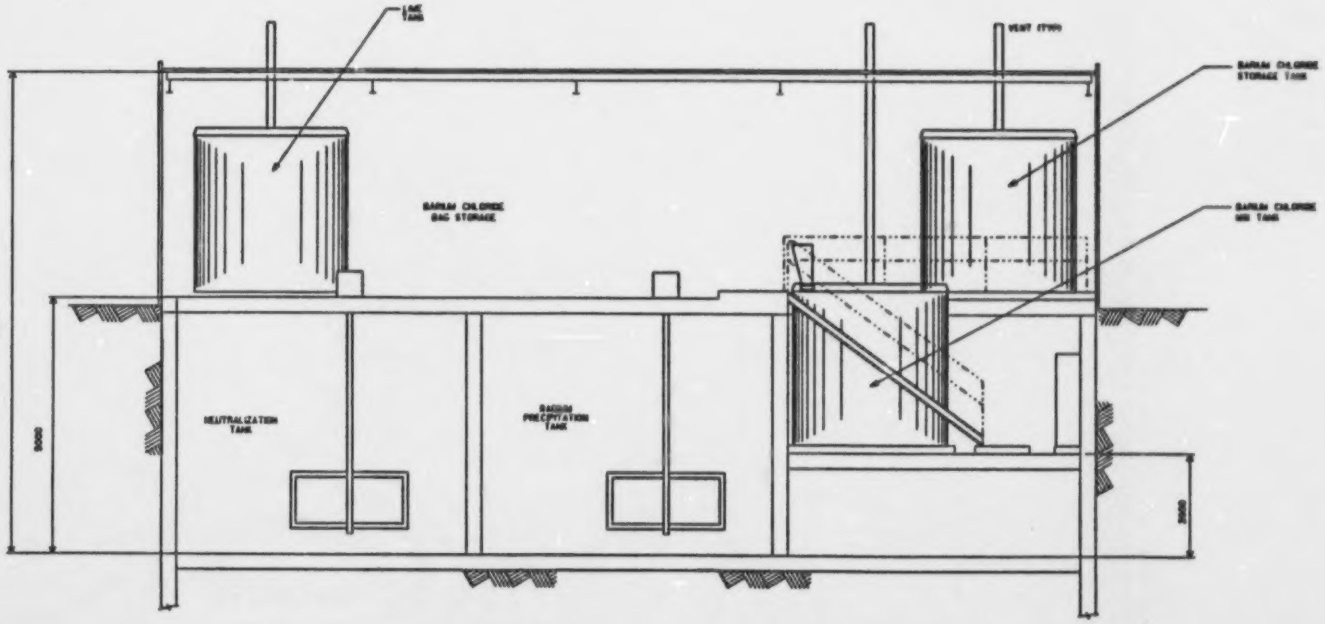
LIME TANK
VENT (TYP)
BARIUM CHLORIDE STORAGE TANK
BARIUM CHLORIDE MIX TANK
BARIUM CHLORIDE BAG STORAGE
NEUTRALIZATION TANK
RADIUM PRECIPITATION TANK
5000
2000

SECTION A-A

NOTE

1. ALL DIMENSIONS ARE IN MILLIMETRES.

0 5000 mm

SCALE=1:50

B RELEASED FOR FINAL REPORT

A RELEASED FOR DRAFT REPORT

GENERAL LAYOUT

1:50

T. WALBAUM

A. LEONARD

SUPPLY AND SERVICES CANADA

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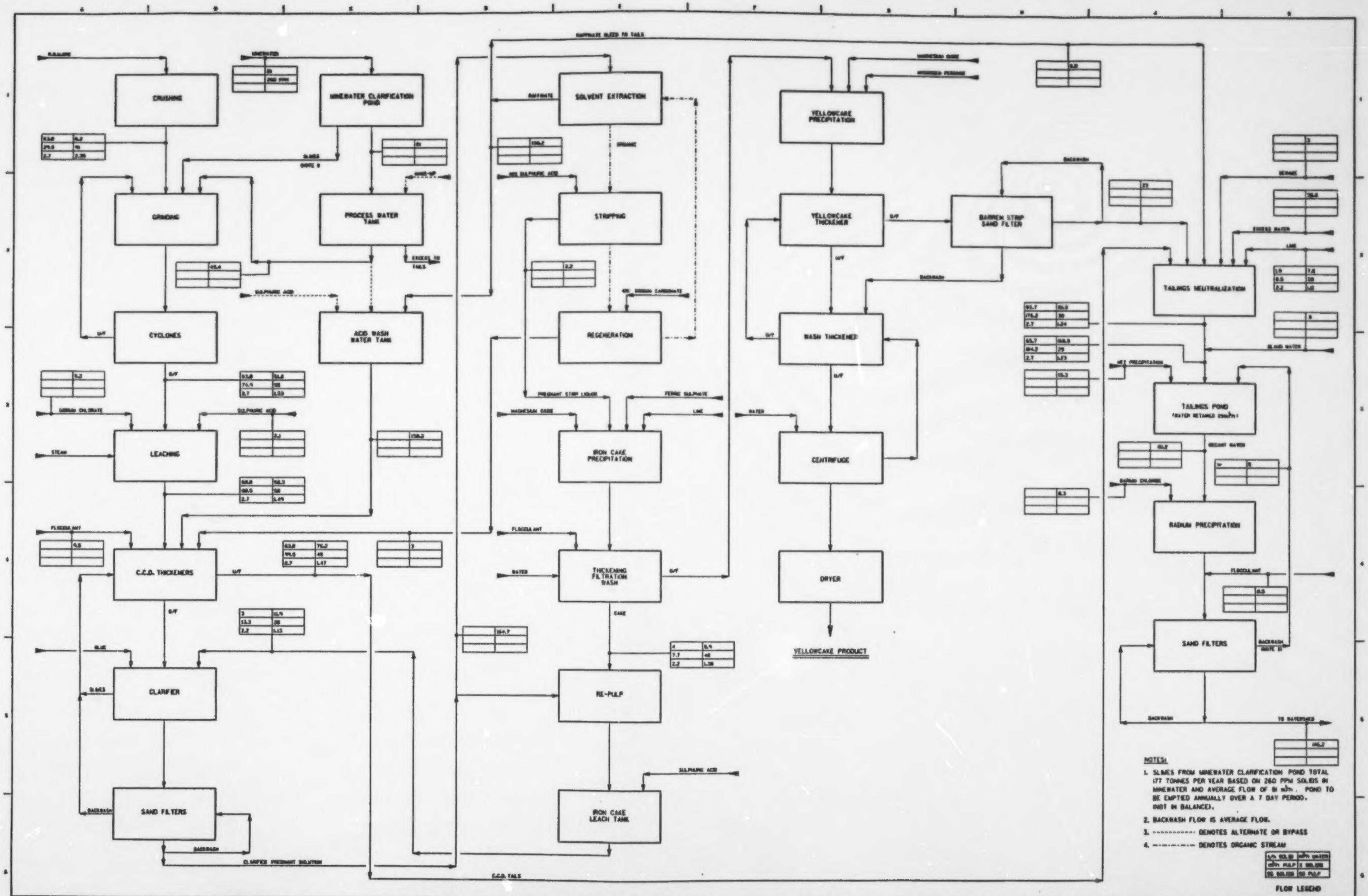
WASTE MANAGEMENT COST STUDY
MODEL A
DECANT TREATMENT BUILDING
PLANS AND SECTION

5239

6

100-10-5

B



NOTES:

1. SLIMES FROM MINEWATER CLARIFICATION POND TOTAL 177 TONNES PER YEAR BASED ON 260 PPM SOLIDS IN MINEWATER AND AVERAGE FLOW OF 81 m³/h. POND TO BE EMPTIED ANNUALLY OVER A 7 DAY PERIOD. (NOT IN BALANCE).
2. BACKWASH FLOW IS AVERAGE FLOW.
3. ------------ DENOTES ALTERNATE OR BYPASS
4. —·—·—·—·— DENOTES ORGANIC STREAM

| | | | |
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| B | RELEASED FOR FINAL REPORT | 1984 | S.R. |
| A | RELEASED FOR DRAFT REPORT | MAR. 12 1984 | CMC |

REVISIONS

REFERENCE DRAWINGS

GENERAL LAYOUT
N.T.S.
L.A.MELIS JAN 1984
R.WARD JAN. 30 1984

SUPPLY AND SERVICES CANADA
OTTAWA, CANADA.

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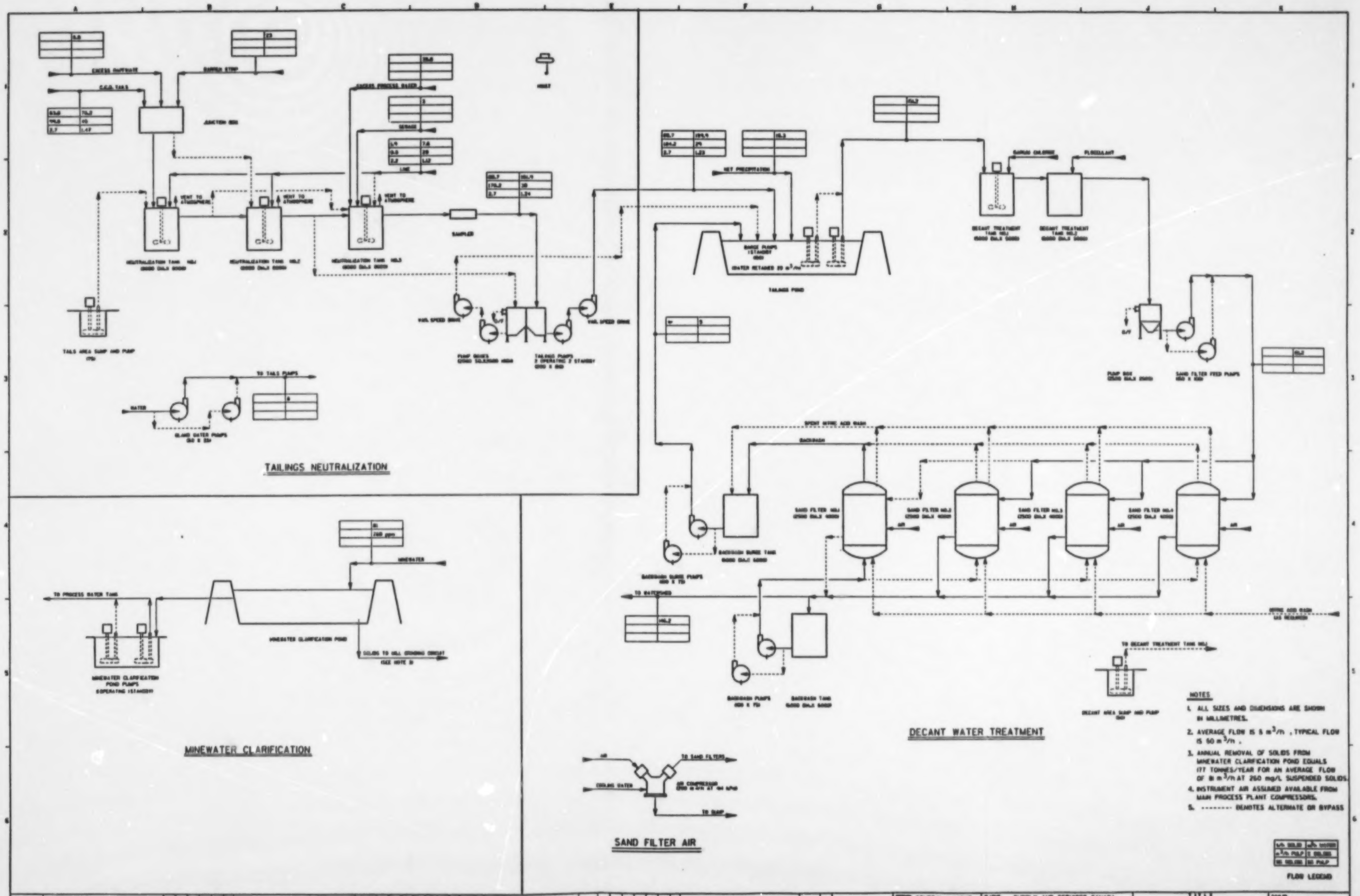
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL B
GENERAL PROCESS FLOWSHEET
BLOCK DIAGRAM

5239 | 15

DRAWING NUMBER
100-10-6 B



NOTES

1. ALL SIZES AND DIMENSIONS ARE SHOWN IN MILLIMETRES.
2. AVERAGE FLOW IS 5 m³/h , TYPICAL FLOW IS 50 m³/h .
3. ANNUAL REMOVAL OF SOLIDS FROM MINEWATER CLARIFICATION POND EQUALS 177 TONNES/YEAR FOR AN AVERAGE FLOW OF 81 m³/h AT 260 mg/L SUSPENDED SOLIDS.
4. INSTRUMENT AIR ASSUMED AVAILABLE FROM MAIN PROCESS PLANT COMPRESSORS.
5. -------- DENOTES ALTERNATE OR BYPASS

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| A | RELEASED FOR DRAFT REPORT | |
| | DESCRIPTION | |

REVISIONS

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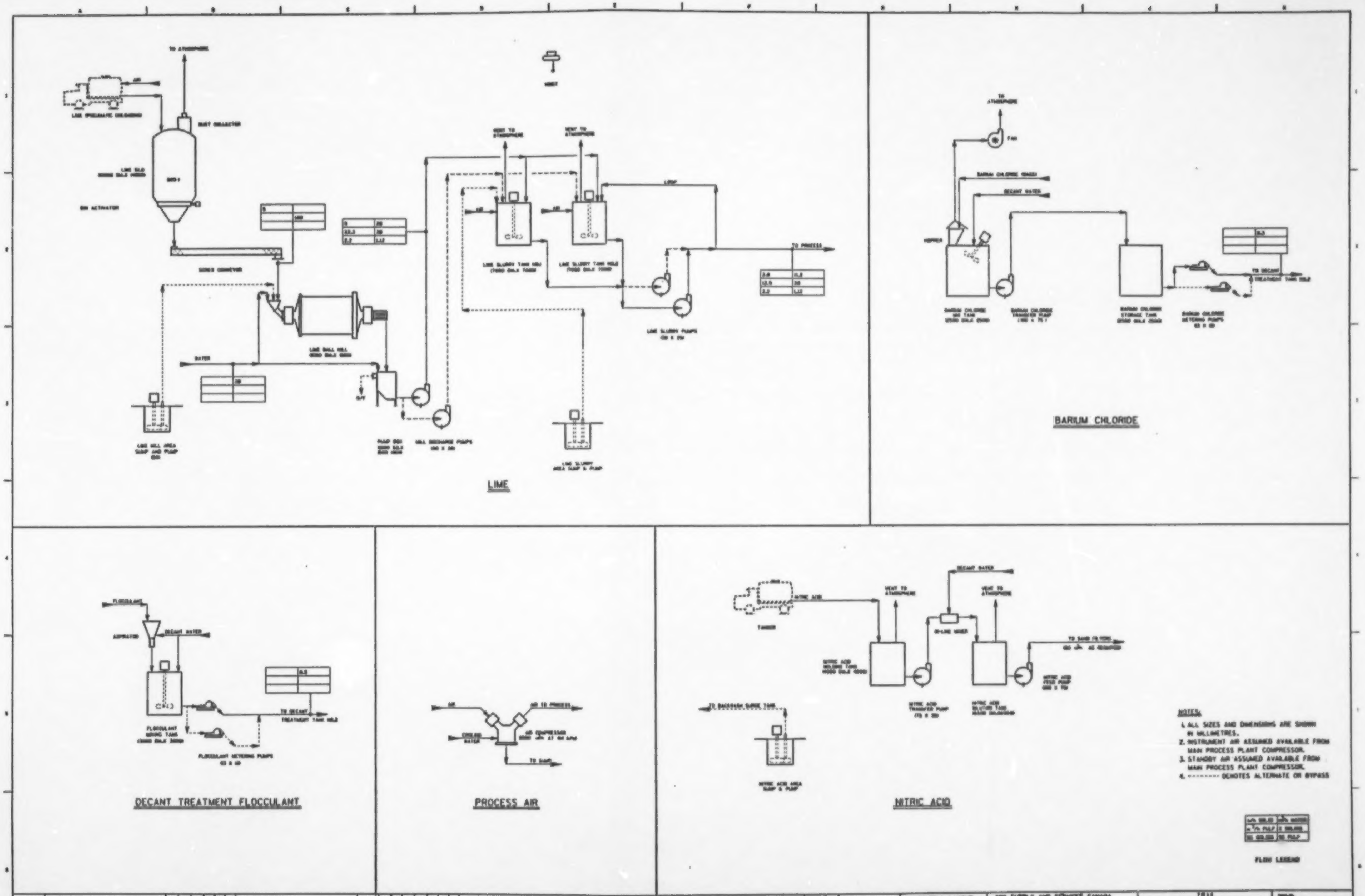
H.T.S.
L.A.MELIS
R.WARD

CLIENT: SUPPLY AND SERVICES CANADA
LOCATION: OTTAWA, CANADA.

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL B
WASTE MANAGEMENT FLOWSHEET

5239
100-10-7
B



NOTES:

1. ALL SIZES AND DIMENSIONS ARE SHOWN IN MILLIMETRES.
2. INSTRUMENT AIR ASSUMED AVAILABLE FROM MAIN PROCESS PLANT COMPRESSOR.
3. STANDBY AIR ASSUMED AVAILABLE FROM MAIN PROCESS PLANT COMPRESSOR.
4. -------- DENOTES ALTERNATE OR BYPASS

| | | |
|---|---|---|
| B | RELEASED FOR FINAL REPORT | |
| A | RELEASED FOR DRAFT REPORT | |

GENERAL LAYOUT
N.T.S.
L.A. NELIS
R. BARD

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OTTAWA, CANADA

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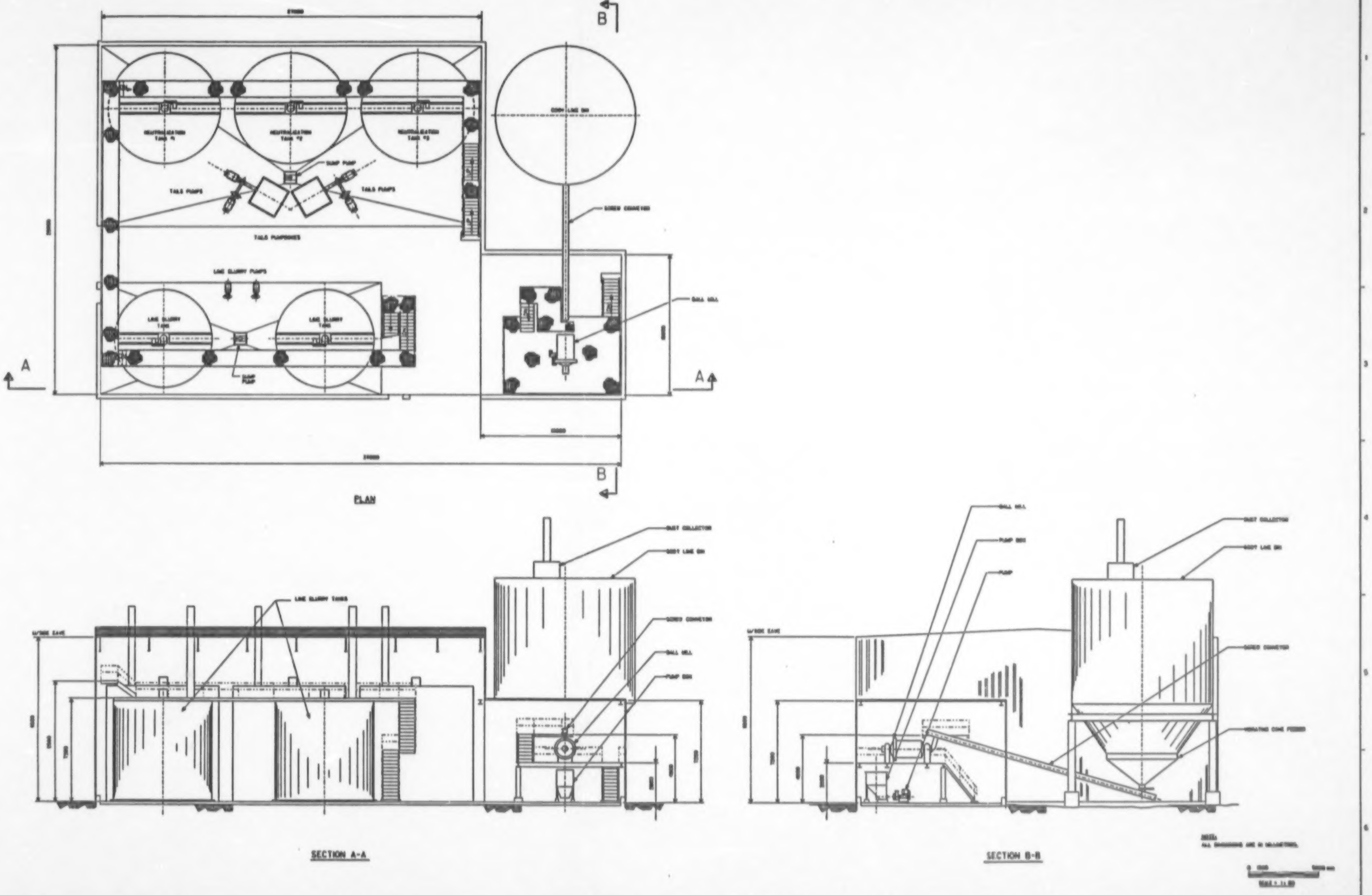
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL B
REAGENT FLOWSHEET

5239
100-10-8
B



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| A | RELEASED FOR DRAFT REPORT | | |
| | DESCRIPTION | | |
| | REVISIONS | | |

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GENERAL LAYOUT

CLIENT: SUPPLY AND SERVICES CANADA

LOCATION: OTTAWA, CANADA

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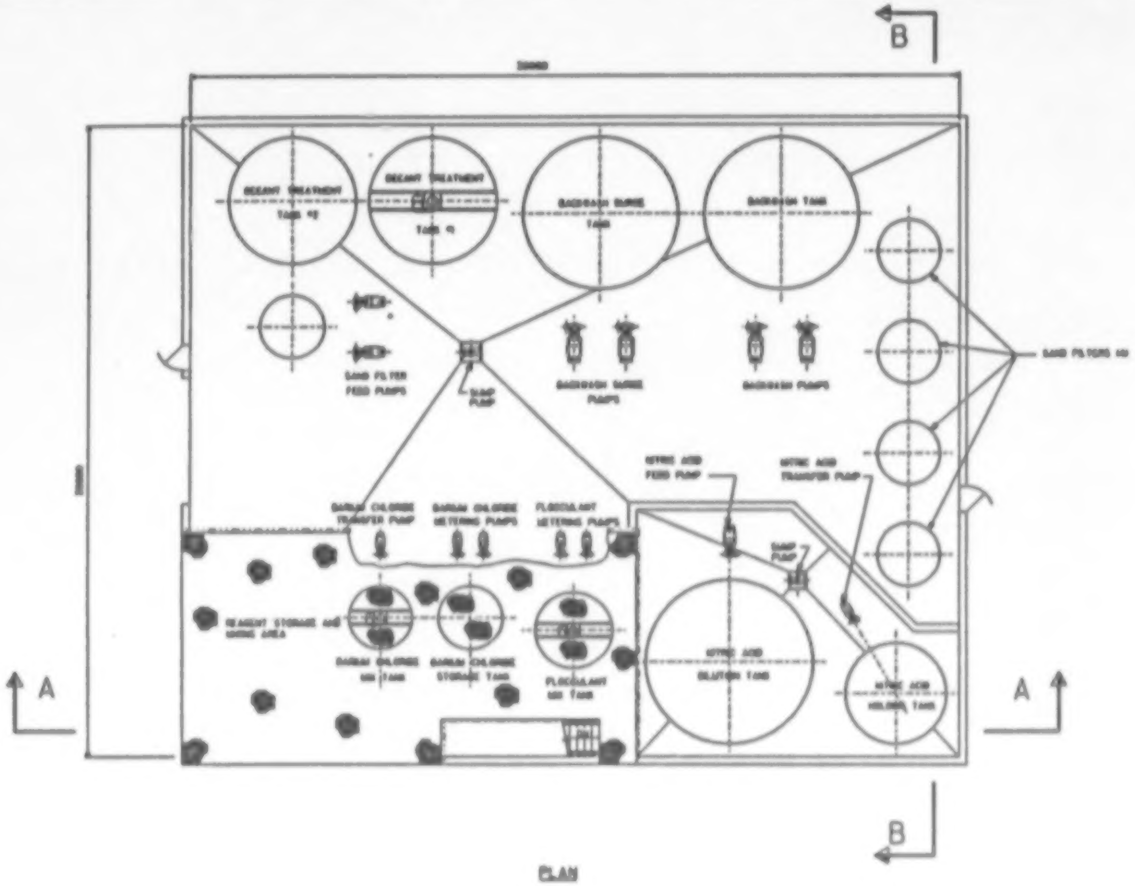
TITLE

NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL B
WASTE TREATMENT BUILDING
PLAN AND SECTIONS

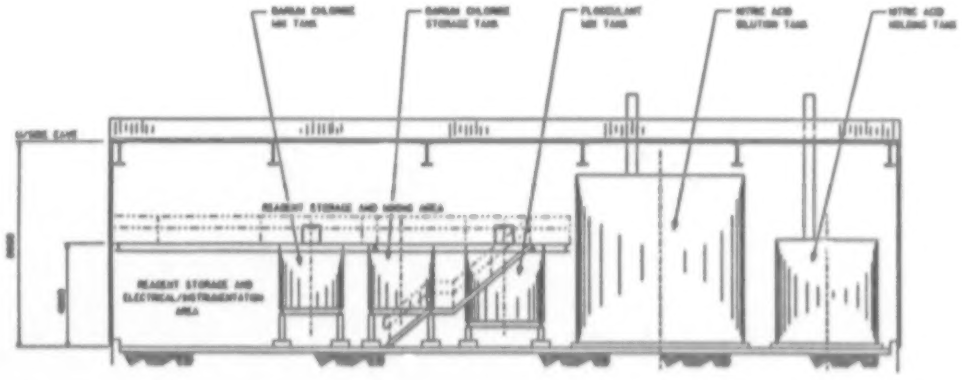
5239

100-10-9

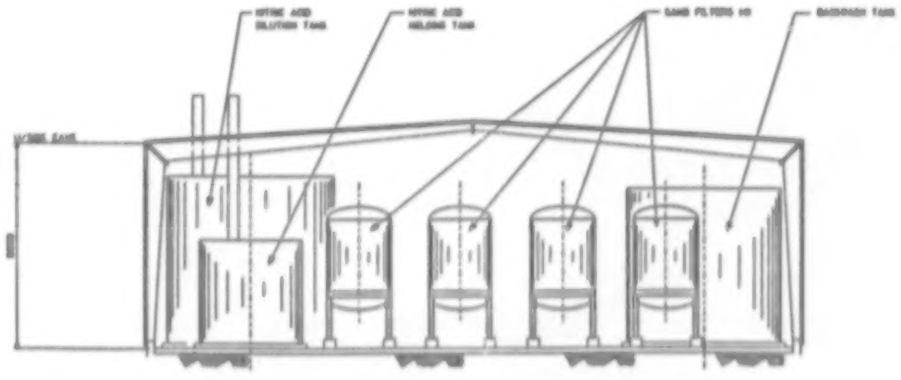
B



PLAN


BARIUM CHLORIDE MIX TANK
BARIUM CHLORIDE STORAGE TANK
FLOCCULANT MIX TANK
NITRIC ACID DILUTION TANK
NITRIC ACID HOLDING TANK
U/SIDE EAVE
REAGENT STORAGE AND MIXING AREA
REAGENT STORAGE AND ELECTRICAL/INSTRUMENTATION AREA

SECTION A-A



SECTION B-B

NOTE:
ALL DIMENSIONS ARE IN MILLIMETRES.

SCALE 1:100

| | | |
|---|---|---|
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| A | RELEASED FOR DRAFT REPORT | |
| No. | DESCRIPTION | DATE |

REVISIONS

REFERENCE DRAWINGS

GENERAL LAYOUT

CLIENT: SUPPLY AND SERVICES CANADA

LOCATION: OTTAWA, CANADA

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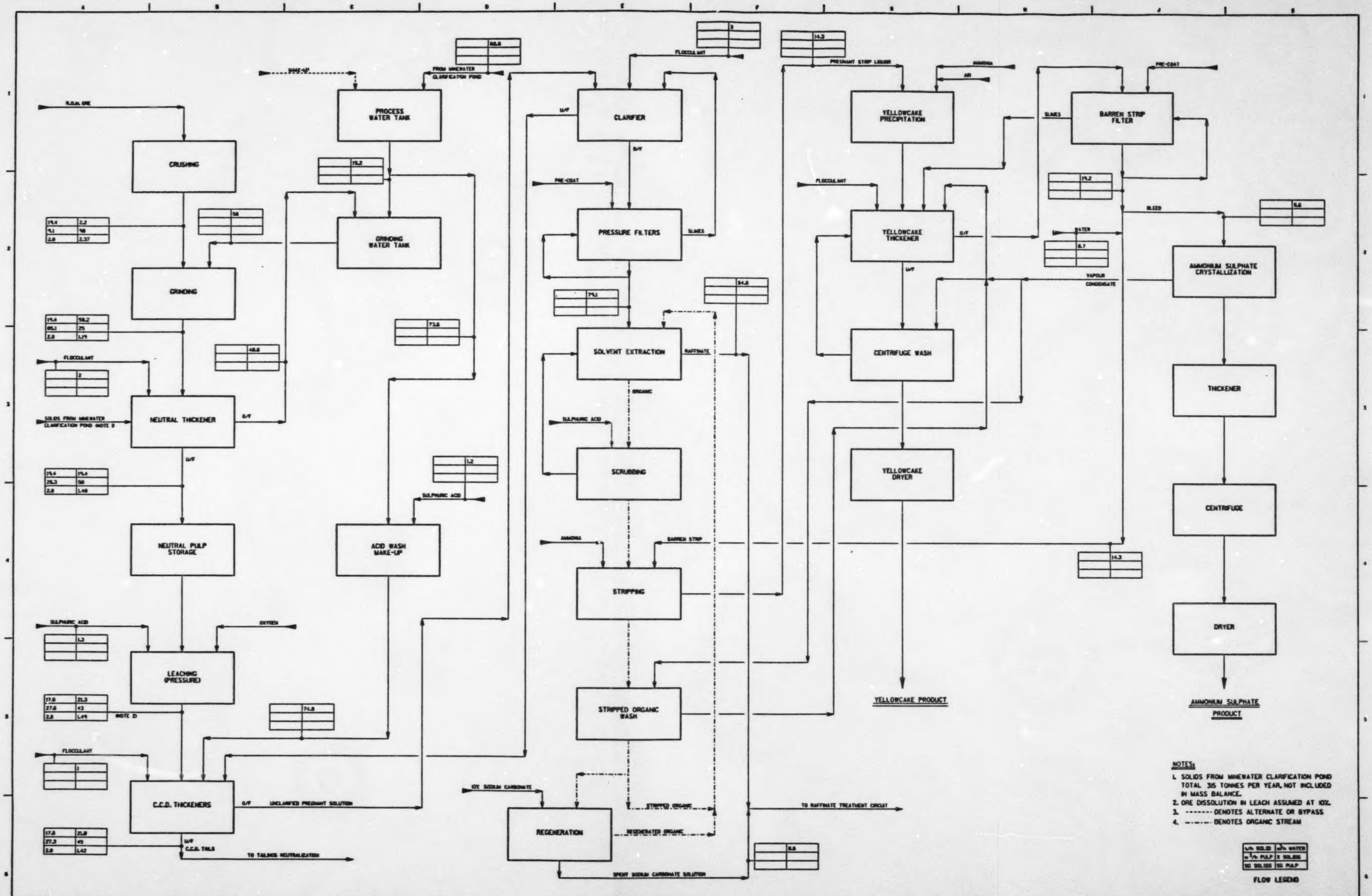
TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL B
DECANT TREATMENT BUILDING
PLAN AND SECTIONS

DRAWING NUMBER
100-10-10
REV. B



NOTES:

1. SOLIDS FROM MINEWATER CLARIFICATION POND TOTAL 315 TONNES PER YEAR, NOT INCLUDED IN MASS BALANCE.
2. ORE DISSOLUTION IN LEACH ASSUMED AT 10%.
3. -------- DENOTES ALTERNATE OR BYPASS
4. —·—·—· DENOTES ORGANIC STREAM

| | |
|---|---|
| t/h SOLID | m³/h WATER |
| m³/h PULP | % SOLIDS |
| SG SOLIDS | SG PULP |

FLOW LEGEND

| | | | |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR. 30 1984 | R.B. |
| A | RELEASED FOR DRAFT REPORT | MAR. 12 1984 | CMC |
| NO. | DESCRIPTION | DATE | BY |

REVISIONS

REFERENCE DRAWINGS

GENERAL LAYOUT
N.T.S.
L.A.MELIS JAN 1984
R.WARD FEB. 3 1984

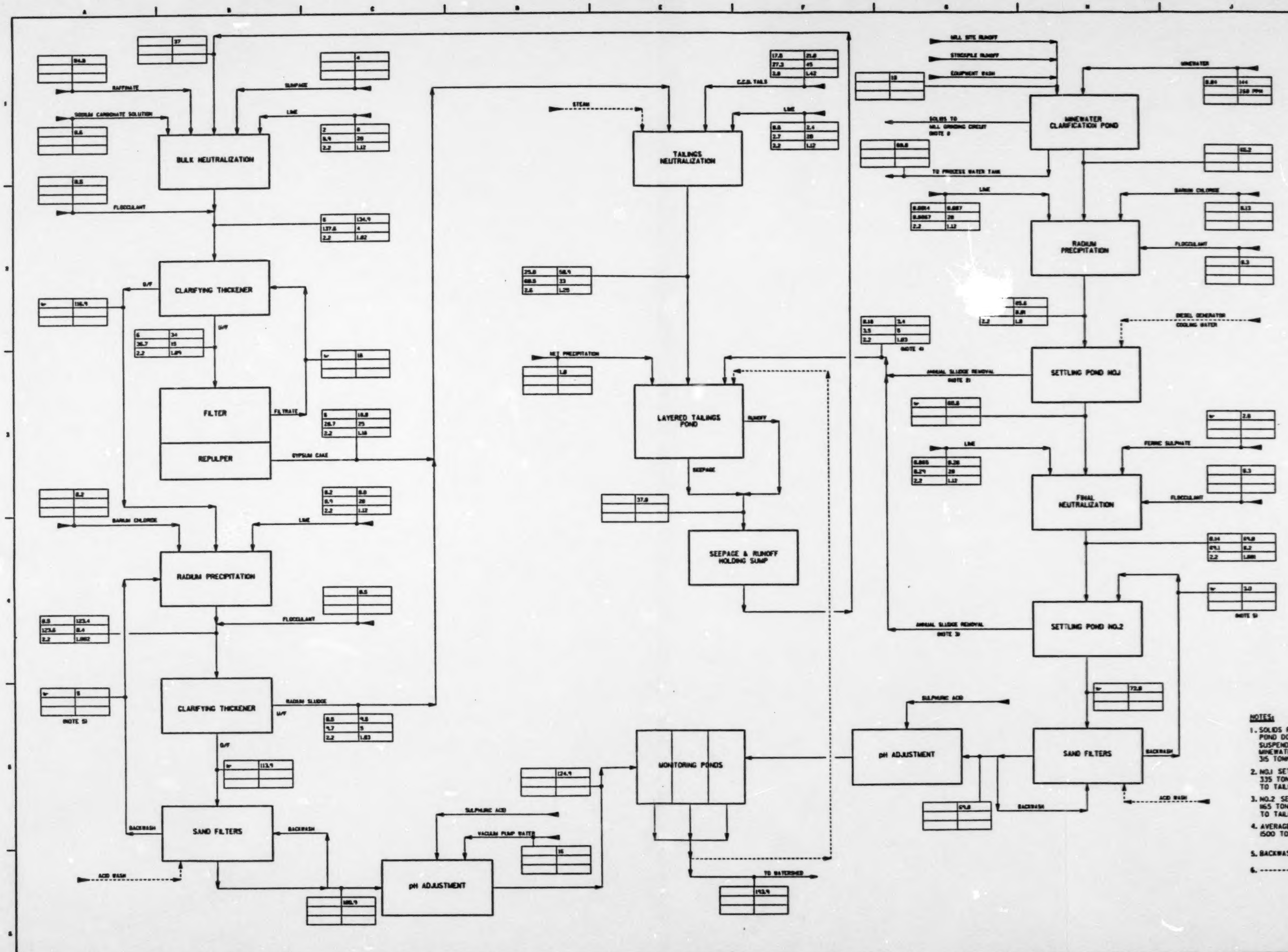
SUPPLY AND SERVICES CANADA
OTTAWA, CANADA.

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
GENERAL PROCESS FLOWSHEET
BLOCK DIAGRAM

5239 | 15

DRAWING NUMBER
100-10-11 B



**NOTES:**

1. SOLIDS REMOVAL FROM MINEWATER STORAGE POND DONE ON AN ANNUAL BASIS. 260 mg/L SUSPENDED SOLIDS IN FEED AND AVERAGE MINEWATER FLOW OF 144 m³/h IS EQUAL TO 315 TONNES PER YEAR OF SOLIDS.
2. NO.1 SETTLING POND SLUDGE ESTIMATED AT 335 TONNES PER YEAR, ANNUAL REMOVAL TO TAILINGS AREA.
3. NO.2 SETTLING POND SLUDGE ESTIMATED AT 165 TONNES PER YEAR, ANNUAL REMOVAL TO TAILINGS AREA.
4. AVERAGE FLOW BASED ON ANNUAL REMOVAL OF 1500 TONNES PER YEAR.
5. BACKWASH FLOW IS AVERAGE FLOW.
6. ---------- DENOTES ALTERNATE OR BYPASS.

| | |
|---|---|
| t/h SOLID | m³/h WATER |
| m³/h PULP | % SOLIDS |
| SG SOLIDS | SG PULP |

FLOW LEGEND

| | | | |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR. 30 1984 | CMC |
| A | RELEASED FOR DRAFT REPORT | MAR. 12 1984 | CMC |
| NO. | DESCRIPTION | DATE | BY |

REVISIONS

REFERENCE DRAWINGS

GENERAL LAYOUT

SCALE N.T.S.

CHECKED L.A.MELIS

DRAWN R.WARD FEB. 9 1984

CLIENT SUPPLY AND SERVICES CANADA

LOCATION OTTAWA, CANADA.

KILBORN

TITLE

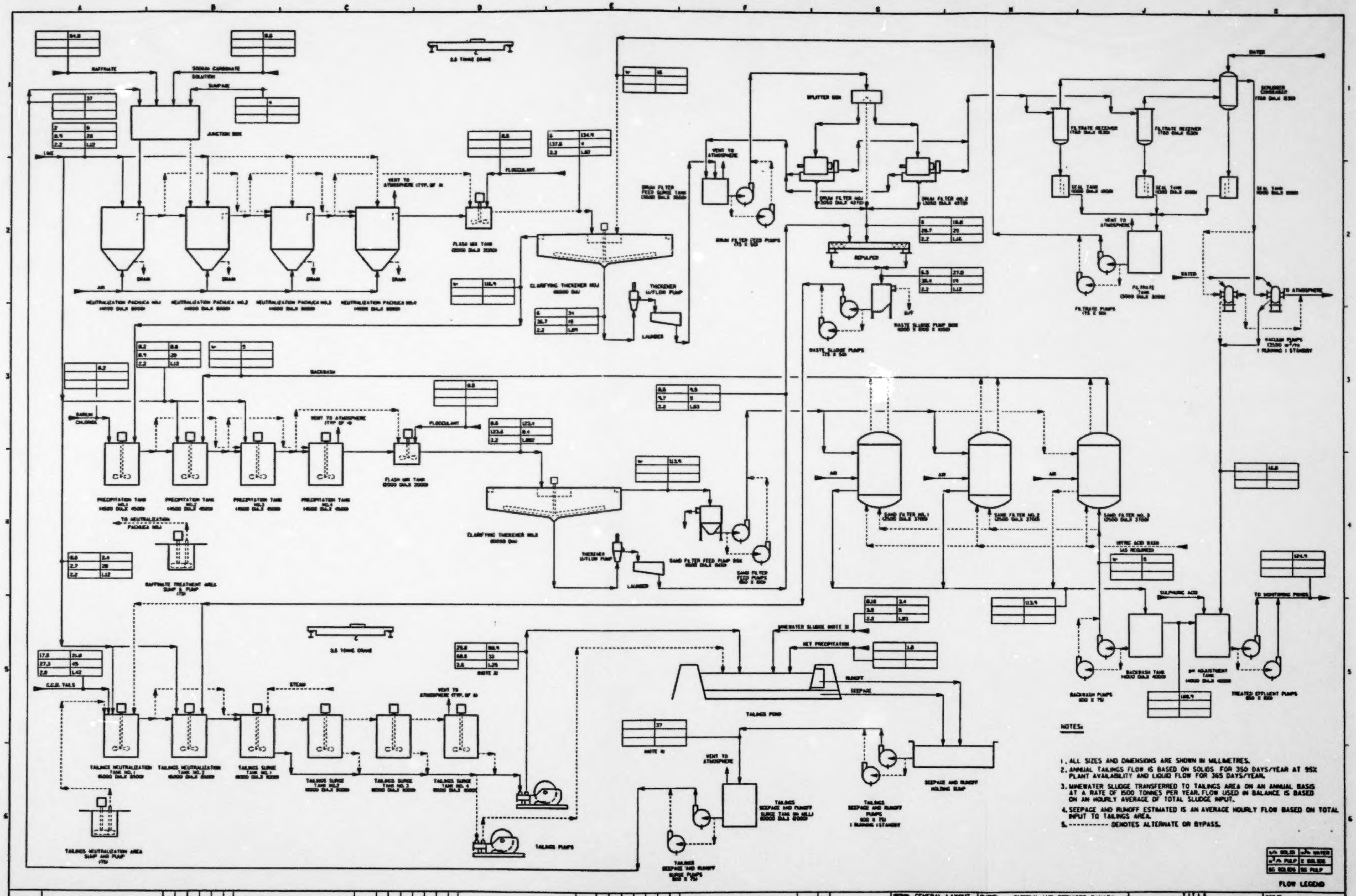
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
WASTE TREATMENT FLOWSHEET
BLOCK DIAGRAM

PROJECT NO. 5239

SHEET NO. 15

DRAWING NUMBER 100-10-12

REV. B


RAFFINATE
SODIUM CARBONATE SOLUTION
SUMPAGE
JUNCTION BOX
LIME
2.5 TONNE CRANE
VENT TO ATMOSPHERE (TYP. OF 4)
FLOCCULANT
NEUTRALIZATION PACHUCA NO.1 (4000 DIA.X 8000)
NEUTRALIZATION PACHUCA NO.2 (4000 DIA.X 8000)
NEUTRALIZATION PACHUCA NO.3 (4000 DIA.X 8000)
NEUTRALIZATION PACHUCA NO.4 (4000 DIA.X 8000)
AIR
DRAIN
FLASH MIX TANK (2000 DIA.X 2000)
CLARIFYING THICKENER NO.1 (6000 DIA)
THICKENER U/FLOW PUMP
LAUNDER
DRUM FILTER FEED SURGE TANK (3500 DIA.X 3500)
VENT TO ATMOSPHERE
DRUM FILTER FEED PUMPS (75 X 50)
SPLITTER BOX
DRUM FILTER NO.1 (3050 DIA.X 4270)
DRUM FILTER NO.2 (3050 DIA.X 4270)
REPULPER
WASTE SLUDGE PUMP BOX (1200 X 1200 X 1200)
WASTE SLUDGE PUMPS (75 X 50)
O/F
FILTRATE RECEIVER (750 DIA.X 1530)
FILTRATE RECEIVER (750 DIA.X 1530)
SCRUBBER CONDENSER (750 DIA.X 1530)
WATER
SEAL TANK (1000 DIA.X 1000)
SEAL TANK (1000 DIA.X 1000)
SEAL TANK (1000 DIA.X 1000)
FILTRATE TANK (3000 DIA.X 3000)
FILTRATE PUMPS (75 X 50)
TO ATMOSPHERE
VACUUM PUMPS (3500 m³/hr) 1 RUNNING 1 STANDBY
BACKWASH
BARIUM CHLORIDE
PRECIPITATION TANK NO.1 (4500 DIA.X 4500)
PRECIPITATION TANK NO.2 (4500 DIA.X 4500)
PRECIPITATION TANK NO.3 (4500 DIA.X 4500)
PRECIPITATION TANK NO.4 (4500 DIA.X 4500)
VENT TO ATMOSPHERE (TYP OF 4)
FLASH MIX TANK (2000 DIA.X 2000)
CLARIFYING THICKENER NO.2 (6000 DIA)
TO NEUTRALIZATION PACHUCA NO.1
RAFFINATE TREATMENT AREA SUMP & PUMP (75)
SAND FILTER FEED PUMP BOX (1500 DIA.X 1500)
SAND FILTER FEED PUMPS (150 X 100)
SAND FILTER NO.1 (2500 DIA.X 3700)
SAND FILTER NO.2 (2500 DIA.X 3700)
SAND FILTER NO.3 (2500 DIA.X 3700)
NITRIC ACID WASH (AS REQUIRED)
SULPHURIC ACID
TO MONITORING PONDS
BACKWASH TANK (4000 DIA.X 4000)
pH ADJUSTMENT TANK (4000 DIA.X 4000)
BACKWASH PUMPS (100 X 75)
TREATED EFFLUENT PUMPS (150 X 100)
C.C.D. TAILS
STEAM
2.5 TONNE CRANE
VENT TO ATMOSPHERE (TYP. OF 6)
TAILINGS NEUTRALIZATION TANK NO.1 (6000 DIA.X 6500)
TAILINGS NEUTRALIZATION TANK NO.2 (6000 DIA.X 6500)
TAILINGS SURGE TANK NO.1 (6000 DIA.X 6000)
TAILINGS SURGE TANK NO.2 (6000 DIA.X 6000)
TAILINGS SURGE TANK NO.3 (6000 DIA.X 6000)
TAILINGS SURGE TANK NO.4 (6000 DIA.X 6000)
TAILINGS NEUTRALIZATION AREA SUMP AND PUMP (75)
TAILINGS PUMPS
(NOTE 3)
MINEWATER SLUDGE (NOTE 3)
NET PRECIPITATION
RUNOFF
SEEPAGE
TAILINGS POND
(NOTE 4)
TAILINGS SEEPAGE AND RUNOFF SURGE TANK ON MILL (6000 DIA.X 6000)
TAILINGS SEEPAGE AND RUNOFF SURGE PUMPS (100 X 75)
TAILINGS SEEPAGE AND RUNOFF PUMPS (100 X 75) 1 RUNNING 1 STANDBY
SEEPAGE AND RUNOFF HOLDING SUMP
84.8
8.6
37
2 | 6
8.9 | 28
2.2 | 1.12
4
8.6
6 | 134.9
137.6 | 4
2.2 | 1.67
16
116.9
6 | 34
36.7 | 15
2.2 | 1.09
6 | 18.8
28.7 | 25
2.2 | 1.16
6.5 | 27.5
30.4 | 19
2.2 | 1.12
8.2 | 8.8
8.9 | 28
2.2 | 1.12
5
8.2
8.6
8.6 | 9.5
9.7 | 5
2.2 | 1.63
8.6 | 123.4
123.6 | 8.4
2.2 | 1.062
113.9
16.8
5
124.9
8.6 | 2.4
2.7 | 28
2.2 | 1.12
8.18 | 3.4
3.5 | 5
2.2 | 1.63
113.9
1.8
168.9
17.5 | 21.8
27.3 | 45
2.8 | 1.42
25.8 | 98.9
98.8 | 33
2.6 | 1.25
37
t/h SOLID | m³/h WATER
m³/h PULP | % SOLIDS
SG SOLIDS | SG PULP
FLOW LEGEND

NOTES:

1. ALL SIZES AND DIMENSIONS ARE SHOWN IN MILLIMETRES.
2. ANNUAL TAILINGS FLOW IS BASED ON SOLIDS FOR 350 DAYS/YEAR AT 95% PLANT AVAILABILITY AND LIQUID FLOW FOR 365 DAYS/YEAR.
3. MINEWATER SLUDGE TRANSFERRED TO TAILINGS AREA ON AN ANNUAL BASIS AT A RATE OF 1500 TONNES PER YEAR. FLOW USED IN BALANCE IS BASED ON AN HOURLY AVERAGE OF TOTAL SLUDGE INPUT.
4. SEEPAGE AND RUNOFF ESTIMATED IS AN AVERAGE HOURLY FLOW BASED ON TOTAL INPUT TO TAILINGS AREA.
5. ---------- DENOTES ALTERNATE OR BYPASS.

| | | | |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR. 30 1984 | CHC |
| A | RELEASED FOR DRAFT REPORT | MAR. 12 1984 | CHC |
| NO. | DESCRIPTION | DATE | BY |
| | REVISIONS | | |

REFERENCE DRAWINGS

GENERAL LAYOUT
N.T.S.
L.A.MELIS
R.WARD FEB. 6 1984

CLIENT: SUPPLY AND SERVICES CANADA

LOCATION: OTTAWA, CANADA.

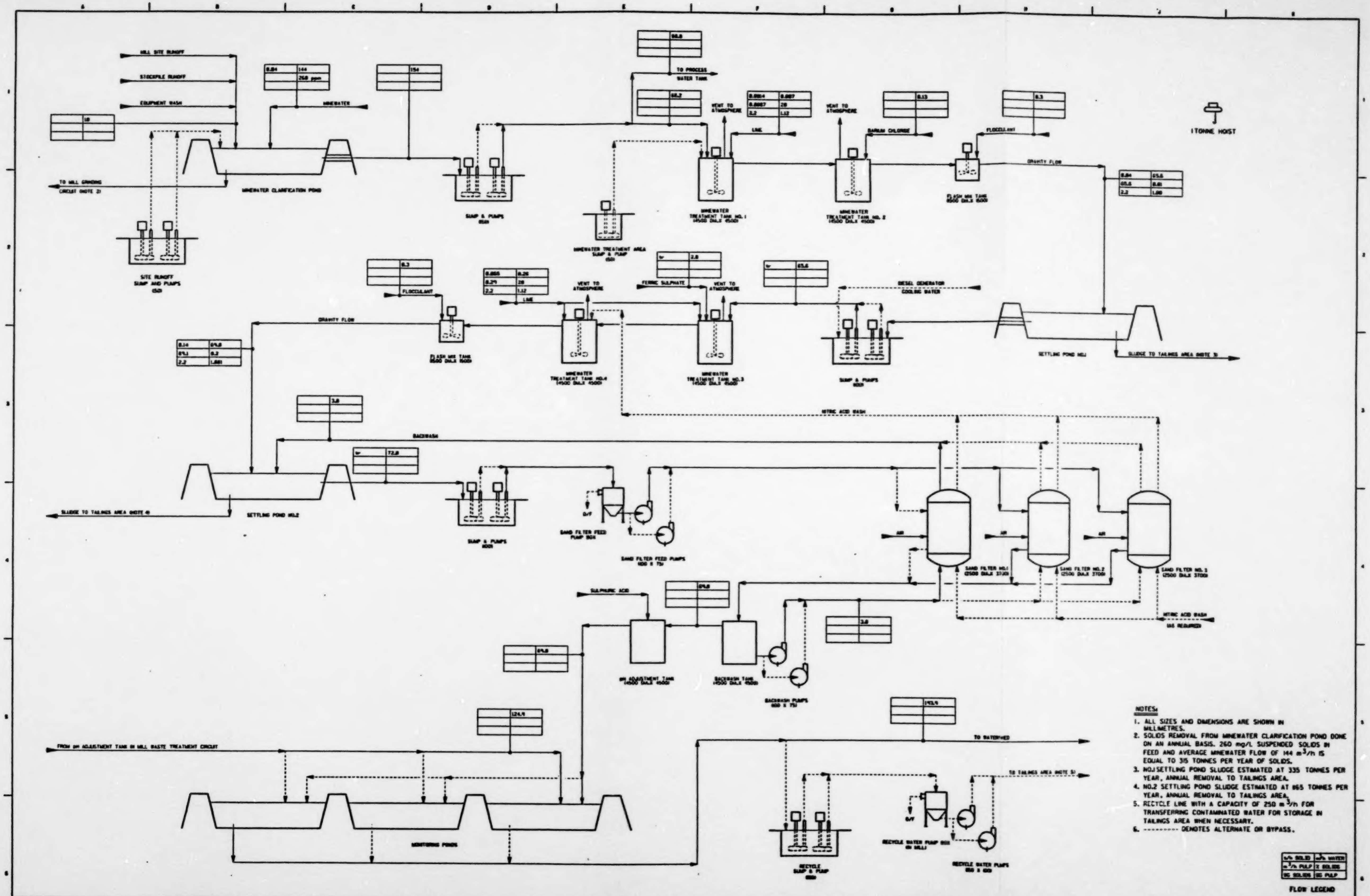
KILBORN

TITLE

NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
MILL WASTE TREATMENT FLOWSHEET

5239 | 15

100-10-13 B


MILL SITE RUNOFF
STOCKPILE RUNOFF
EQUIPMENT WASH
MINEWATER
10
0.84 | 144
260 ppm
154
TO MILL GRINDING CIRCUIT (NOTE 2)
MINEWATER CLARIFICATION POND
SITE RUNOFF SUMP AND PUMPS (50)
SUMP & PUMPS (150)
98.8
TO PROCESS WATER TANK
58.2
VENT TO ATMOSPHERE
0.0014 | 0.087
0.0087 | 28
2.2 | 1.12
LIME
MINEWATER TREATMENT AREA SUMP & PUMP (50)
MINEWATER TREATMENT TANK NO.1 (4500 DIA.X 4500)
VENT TO ATMOSPHERE
0.13
BARIUM CHLORIDE
MINEWATER TREATMENT TANK NO.2 (4500 DIA.X 4500)
0.3
FLOCCULANT
FLASH MIX TANK (1500 DIA.X 1500)
GRAVITY FLOW
0.84 | 65.6
65.6 | 0.61
2.2 | 1.00
1 TONNE HOIST
0.3
FLOCCULANT
0.055 | 0.26
0.29 | 28
2.2 | 1.12
LIME
VENT TO ATMOSPHERE
FERRIC SULPHATE
2.8
VENT TO ATMOSPHERE
65.6
DIESEL GENERATOR COOLING WATER
GRAVITY FLOW
FLASH MIX TANK (1500 DIA.X 1500)
MINEWATER TREATMENT TANK NO.4 (4500 DIA.X 4500)
MINEWATER TREATMENT TANK NO.3 (4500 DIA.X 4500)
SUMP & PUMPS (100)
SETTLING POND NO.1
SLUDGE TO TAILINGS AREA (NOTE 3)
0.14 | 69.8
69.1 | 0.2
2.2 | 1.001
NITRIC ACID WASH
3.0
BACKWASH
72.8
SLUDGE TO TAILINGS AREA (NOTE 4)
SETTLING POND NO.2
SUMP & PUMPS (100)
O/F
SAND FILTER FEED PUMP BOX
SAND FILTER FEED PUMPS (100 X 75)
AIR
AIR
AIR
SAND FILTER NO.1 (2500 DIA.X 3700)
SAND FILTER NO.2 (2500 DIA.X 3700)
SAND FILTER NO.3 (2500 DIA.X 3700)
NITRIC ACID WASH (AS REQUIRED)
SULPHURIC ACID
69.8
3.0
69.8
pH ADJUSTMENT TANK (4500 DIA.X 4500)
BACKWASH TANK (4500 DIA.X 4500)
BACKWASH PUMPS (100 X 75)
124.9
193.9
FROM pH ADJUSTMENT TANK IN MILL WASTE TREATMENT CIRCUIT
TO WATERSHED
TO TAILINGS AREA (NOTE 5)
MONITORING PONDS
RECYCLE SUMP & PUMP
O/F
RECYCLE WATER PUMP BOX (IN MILL)
RECYCLE WATER PUMPS (150 X 100)

**NOTES:**

1. ALL SIZES AND DIMENSIONS ARE SHOWN IN MILLIMETRES.
2. SOLIDS REMOVAL FROM MINEWATER CLARIFICATION POND DONE ON AN ANNUAL BASIS. 260 mg/L SUSPENDED SOLIDS IN FEED AND AVERAGE MINEWATER FLOW OF 144 $m^3/h$ IS EQUAL TO 315 TONNES PER YEAR OF SOLIDS.
3. NO.1 SETTLING POND SLUDGE ESTIMATED AT 335 TONNES PER YEAR, ANNUAL REMOVAL TO TAILINGS AREA.
4. NO.2 SETTLING POND SLUDGE ESTIMATED AT 865 TONNES PER YEAR, ANNUAL REMOVAL TO TAILINGS AREA.
5. RECYCLE LINE WITH A CAPACITY OF 250 $m^3/h$ FOR TRANSFERRING CONTAMINATED WATER FOR STORAGE IN TAILINGS AREA WHEN NECESSARY.
6. --------- DENOTES ALTERNATE OR BYPASS.

| | |
|---|---|
| t/h SOLID | $m^3/h$ WATER |
| $m^3/h$ PULP | % SOLIDS |
| SG SOLIDS | SG PULP |

FLOW LEGEND

| | | | |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR.30 1984 | CHC |
| A | RELEASED FOR DRAFT REPORT | MAR.12 1984 | CHC |
| | DESCRIPTION | DATE | BY |

REVISIONS

REFERENCE DRAWINGS

GENERAL LAYOUT
N.T.S.
L.A.MELIS
R.WARD — FEB.17 1984

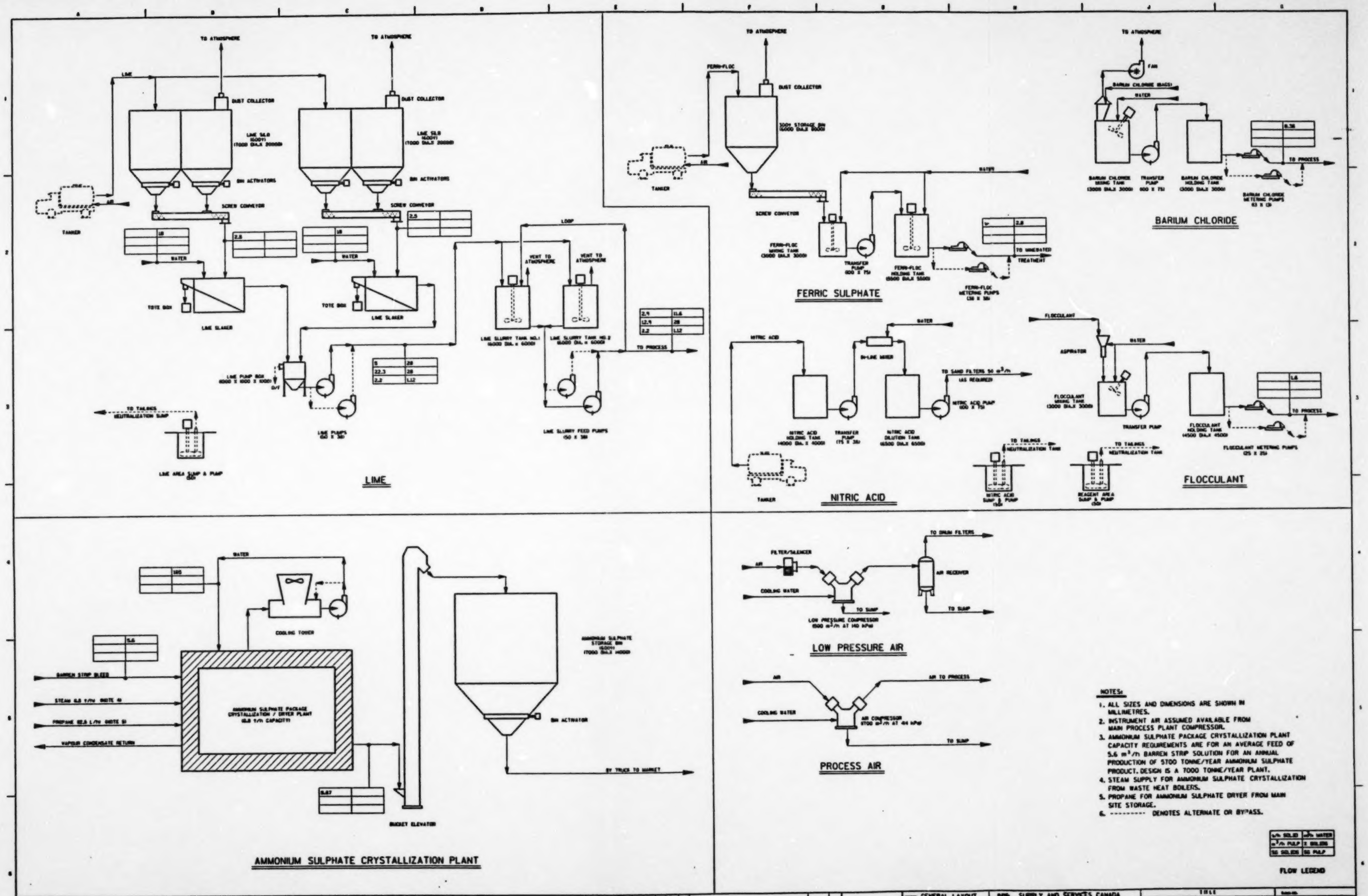
SUPPLY & SERVICES CANADA
OTTAWA, CANADA.

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
MINEWATER TREATMENT FLOWSHEET

5239 | 15

100-10-14 B


TO ATMOSPHERE
LIME
DUST COLLECTOR
LIME SILO (600T) (7000 DIA.X 20000)
BIN ACTIVATORS
SCREW CONVEYOR
TANKER
AIR
18
2.5
WATER
TOTE BOX
LIME SLAKER
LOOP
VENT TO ATMOSPHERE
LIME SLURRY TANK NO.1 (6000 DIA. x 6000)
LIME SLURRY TANK NO.2 (6000 DIA. x 6000)
LIME PUMP BOX (6000 X 1000 X 1000)
O/F
5
28
22.3
2.2
1.12
2.9
11.6
12.9
TO PROCESS
LIME PUMPS (50 X 38)
LIME SLURRY FEED PUMPS (50 X 38)
TO TAILINGS NEUTRALIZATION SUMP
LIME AREA SUMP & PUMP (50)
LIME
FERRI-FLOC
300t STORAGE BIN (6000 DIA.X 9000)
SCREW CONVEYOR
FERRI-FLOC MIXING TANK (3000 DIA.X 3000)
TRANSFER PUMP (100 X 75)
FERRI-FLOC HOLDING TANK (5500 DIA.X 5500)
FERRI-FLOC METERING PUMPS (38 X 38)
2.8
TO UNIDENTIFIED TREATMENT
FERRIC SULPHATE
FAN
BARIUM CHLORIDE (BAGS)
BARIUM CHLORIDE MIXING TANK (3000 DIA.X 3000)
TRANSFER PUMP (100 X 75)
BARIUM CHLORIDE HOLDING TANK (3000 DIA.X 3000)
BARIUM CHLORIDE METERING PUMPS (13 X 13)
0.36
BARIUM CHLORIDE
NITRIC ACID
IN-LINE MIXER
NITRIC ACID HOLDING TANK (4000 DIA.X 4000)
TRANSFER PUMP (75 X 38)
NITRIC ACID DILUTION TANK (6500 DIA.X 6500)
NITRIC ACID PUMP (100 X 75)
TO SAND FILTERS 50 m³/h (AS REQUIRED)
TO TAILINGS NEUTRALIZATION TANK
NITRIC ACID SUMP & PUMP (50)
NITRIC ACID
FLOCCULANT
ASPIRATOR
FLOCCULANT MIXING TANK (3000 DIA.X 3000)
TRANSFER PUMP
FLOCCULANT HOLDING TANK (4500 DIA.X 4500)
FLOCCULANT METERING PUMPS (25 X 25)
1.6
REAGENT AREA SUMP & PUMP (50)
FLOCCULANT
WATER
105
COOLING TOWER
5.6
BARREN STRIP BLEED
STEAM 6.8 T/H (NOTE 4)
PROPANE 82.5 L/H (NOTE 5)
VAPOUR CONDENSATE RETURN
AMMONIUM SULPHATE PACKAGE CRYSTALLIZATION / DRYER PLANT (0.8 T/H CAPACITY)
0.87
BUCKET ELEVATOR
AMMONIUM SULPHATE STORAGE BIN (600T) (7000 DIA.X 14000)
BIN ACTIVATOR
BY TRUCK TO MARKET
AMMONIUM SULPHATE CRYSTALLIZATION PLANT
FILTER/SILENCER
AIR
COOLING WATER
TO SUMP
LOW PRESSURE COMPRESSOR 1500 m³/h AT 140 kPa
AIR RECEIVER
TO DRUM FILTERS
LOW PRESSURE AIR
AIR TO PROCESS
AIR COMPRESSOR 1700 m³/h AT 414 kPa
PROCESS AIR
t/h SOLID
m³/h WATER
m³/h PULP
% SOLIDS
SG SOLIDS
SG PULP
FLOW LEGEND

NOTES:

1. ALL SIZES AND DIMENSIONS ARE SHOWN IN MILLIMETRES.
2. INSTRUMENT AIR ASSUMED AVAILABLE FROM MAIN PROCESS PLANT COMPRESSOR.
3. AMMONIUM SULPHATE PACKAGE CRYSTALLIZATION PLANT CAPACITY REQUIREMENTS ARE FOR AN AVERAGE FEED OF 5.6 $m^3/h$ BARREN STRIP SOLUTION FOR AN ANNUAL PRODUCTION OF 5700 TONNE/YEAR AMMONIUM SULPHATE PRODUCT. DESIGN IS A 7000 TONNE/YEAR PLANT.
4. STEAM SUPPLY FOR AMMONIUM SULPHATE CRYSTALLIZATION FROM WASTE HEAT BOILERS.
5. PROPANE FOR AMMONIUM SULPHATE DRYER FROM MAIN SITE STORAGE.
6. --------- DENOTES ALTERNATE OR BYPASS.

| | | | | |
|---|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | | | |
| A | RELEASED FOR DRAFT REPORT | | | |
| | DESCRIPTION | | | |
| | REVISIONS | | | |

REFERENCE DRAWINGS

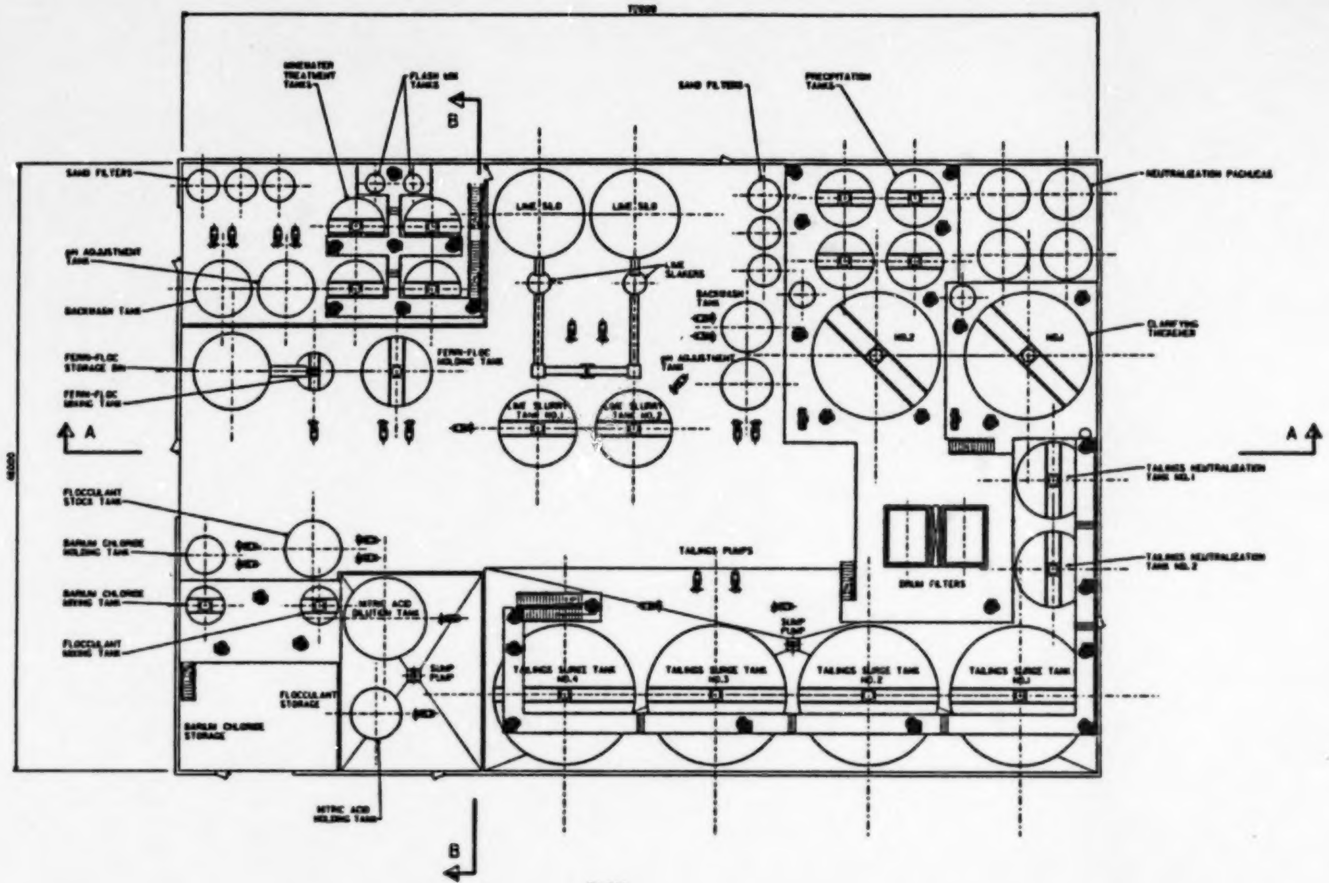
GENERAL LAYOUT
N.T.S.
L.A.MELIS
A.EDMONSTONE

SUPPLY AND SERVICES CANADA
OTTAWA, CANADA

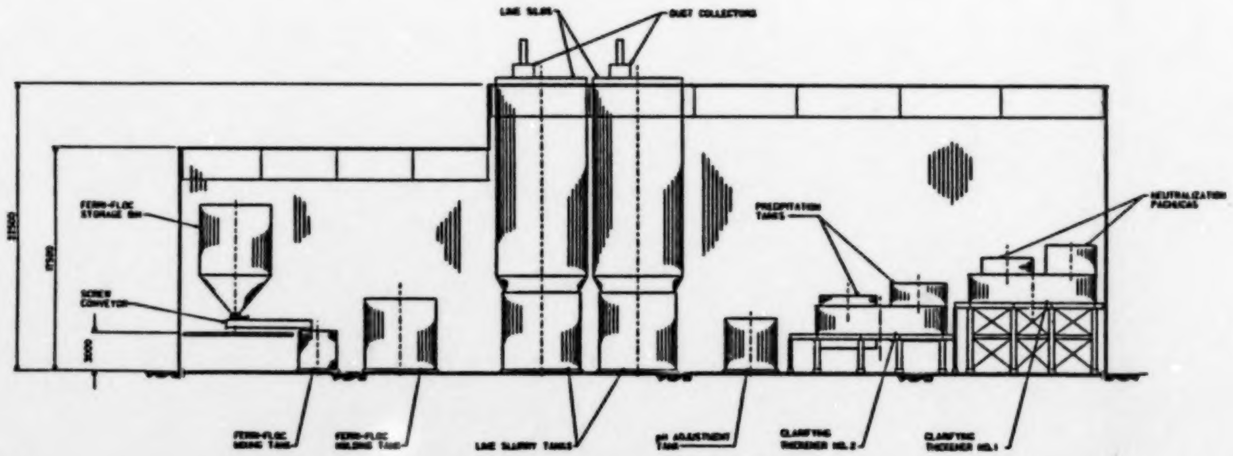
KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
REAGENTS AND UTILITIES FLOWSHEET

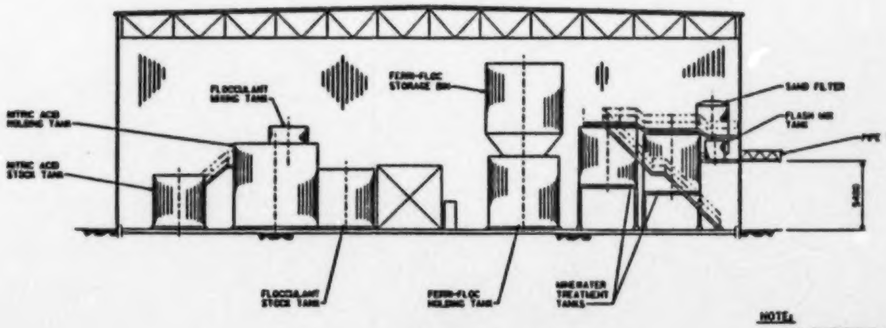
5239 | 15
DRAWING NUMBER
100-10-15 | B



PLAN



SECTION A-A



SECTION B-B

NOTE:
-ALL DIMENSIONS ARE IN MILLIMETRES

0 2000 10000 18000mm

REFERENCE DRAWINGS

B RELEASED FOR FINAL REPORT CMC
A RELEASED FOR DRAFT REPORT MAR. 12 1984 CMC

REVISIONS

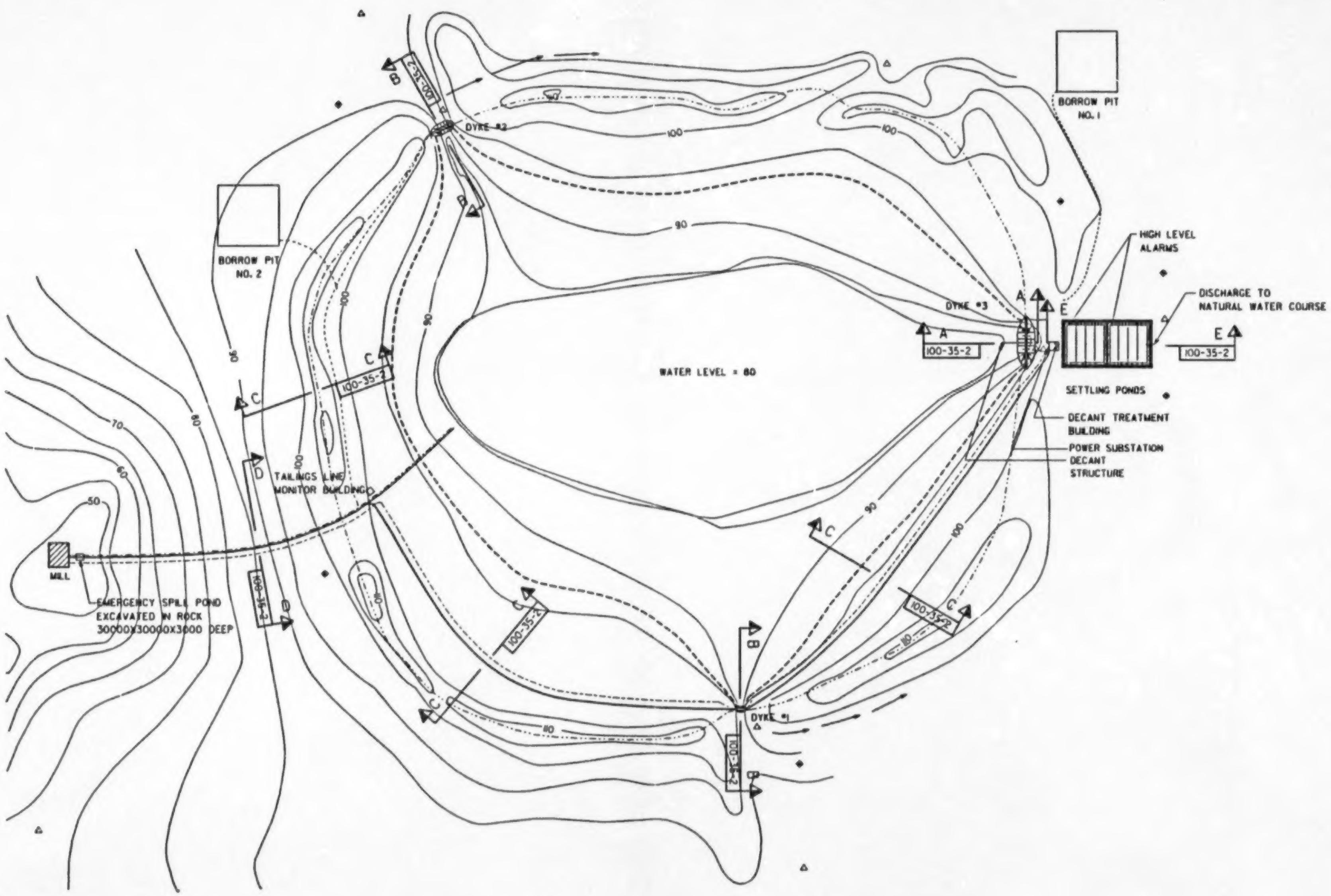
GENERAL LAYOUT
1:200
R.WARD FEB 27 1984
R.WARD FEB 27 1984

SUPPLY & SERVICES CANADA
OTTAWA, CANADA.

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
WASTE TREATMENT BUILDING
PLAN AND SECTIONS

5239 15
100-10-16 B



**LEGEND**

| | |
|---|---|
| CONTOUR | 100 |
| PERMANENT ROAD | ─── |
| TEMPORARY CONSTRUCTION ROAD | ┈┈┈ |
| FINAL TAILINGS LEVEL | ╌╌╌ |
| TAILINGS PIPELINE | ─·─·─ |
| DRAINAGE BASIN | ─··─··─ |
| POWER SUPPLY LINE - 3 PHASE | ─ ─ ─ |
| FRESH WATER DIVERSION DITCHES | ──→ ──→ |
| SURFACE WATER MONITORING STATIONS | △ |
| GROUNDWATER MONITORING PIEZOMETER | ◆ |

**NOTE**

1) ALL ELEVATIONS ARE IN METRES.
2) ASSUMED MILL ELEVATION 50.0
3) ALL DIMENSIONS ARE IN MILLIMETRES.
4) CLEARING REQUIRED; ABOVE LAKE LEVEL TO FINAL TAILINGS LEVEL, DYKES, SETTLING PONDS, BORROW PITS.

0 100000 500000 1000000mm

| DWG. NO. | DESCRIPTION |
|---|---|
| 100-35-2 | DETAILS |

REFERENCE DRAWINGS

| NO. | DESCRIPTION | DATE | BY |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | 30 MAR 1984 | RW |
| A | RELEASED FOR DRAFT REPORT | 12 MAR 1984 | PF |

REVISIONS

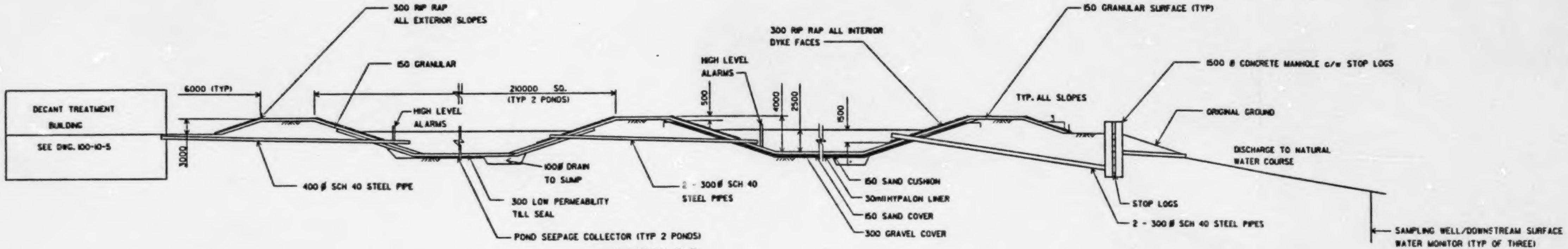
CIVIL LAYOUT
SCALE 1:10000
DESIGNED T.H. WALBAUM JAN 1984
DRAWN P.A. FAHIE 20 JAN 1984

CLIENT: SUPPLY AND SERVICES CANADA
LOCATION: OTTAWA, CANADA

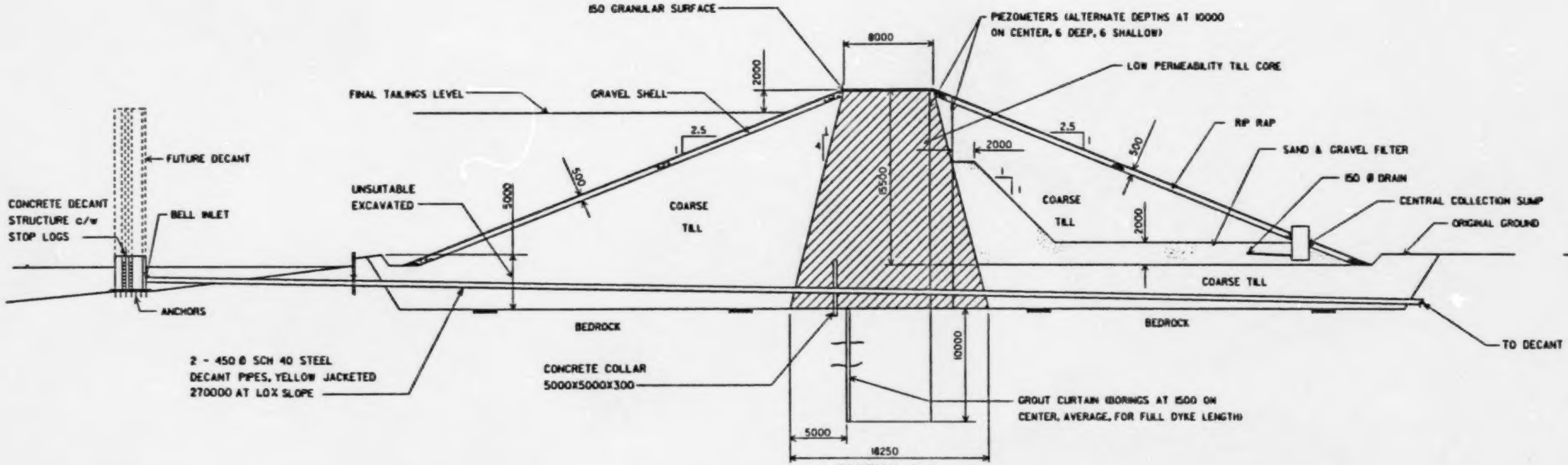
KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL A
WASTE MANAGEMENT AREA
SITE PLAN

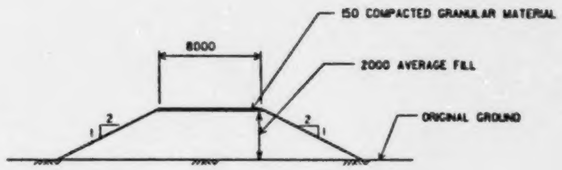
PROJECT NO. 5239
DIVISION NO. 15
DRAWING NUMBER 100-35-1
REV. B



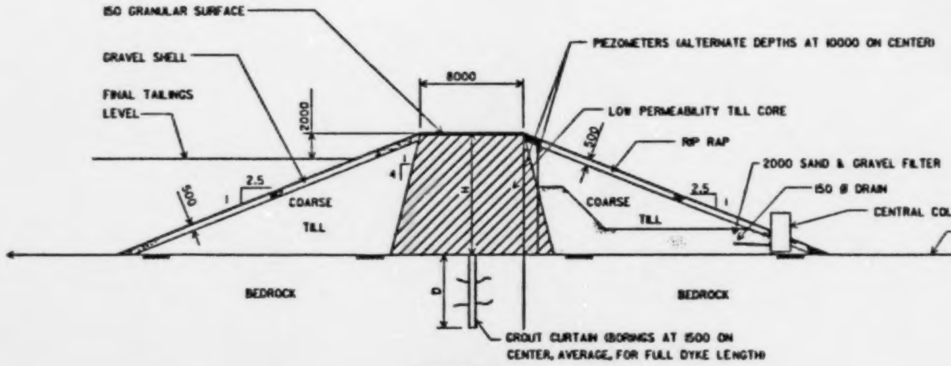
SECTION E-E
(100-35-1)



SECTION A-A
(100-35-1)

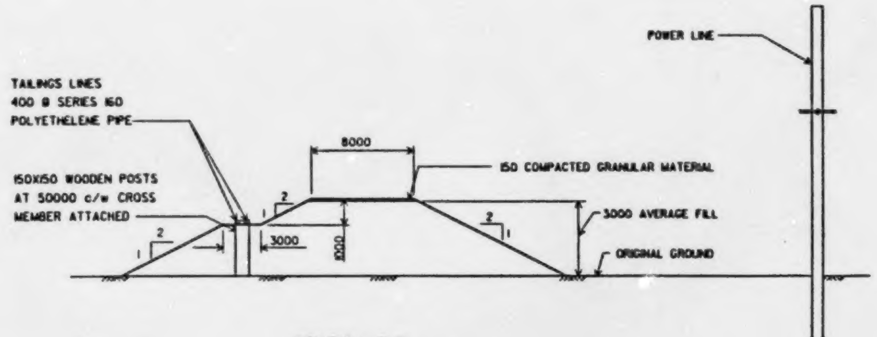


SECTION C-C
(100-35-1)


150 GRANULAR SURFACE
PIEZOMETERS (ALTERNATE DEPTHS AT 10000 ON CENTER)
GRAVEL SHELL
8000
FINAL TAILINGS
LEVEL
2000
LOW PERMEABILITY TILL CORE
500
RIP RAP
2000 SAND & GRAVEL FILTER
150 Ø DRAIN
CENTRAL COLLECTION SUMP
ORIGINAL GROUND
500
2.5
1
4
1
COARSE
TILL
COARSE
TILL
2.5
1
H
BEDROCK
BEDROCK
D
GROUT CURTAIN (BORINGS AT 1500 ON
CENTER, AVERAGE, FOR FULL DYKE LENGTH)

| NOTE | | | PIEZOMETERS |
|---|---|---|---|
| | DYKE #1 | H=4500 D=3000 | 1 DEEP, 1 SHALLOW |
| | DYKE #2 | H=9500 D=5000 | 3 DEEP, 3 SHALLOW |

SECTION B-B
(100-35-1)



SECTION D-D
(100-35-1)
SCALE 1:200

NOTE
ALL DIMENSIONS ARE IN MILLIMETRES.

0 5000 10000 20000mm

| REF. NO. | DESCRIPTION |
|---|---|
| 100-35-1 | SITE PLAN |
| | REFERENCE DRAWINGS |

| NO. | DESCRIPTION | DATE | BY |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR 30 1984 | R.W. |
| A | RELEASED FOR DRAFT REPORT | MAR 12 1984 | E.H. |
| | REVISIONS | | |

CIVIL LAYOUT
1:200
T.H.WALBAUM
P.A. FAHIE

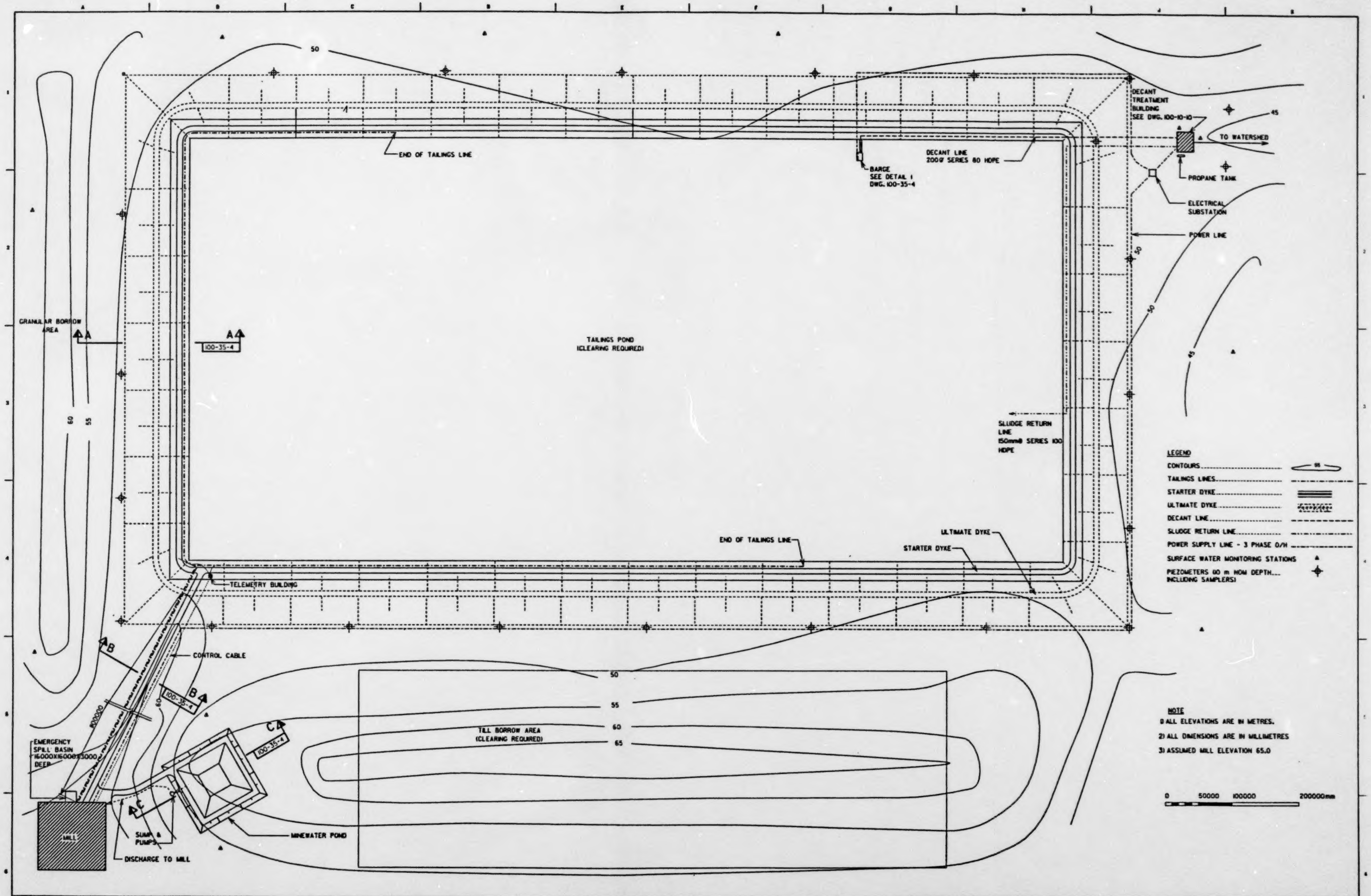
SUPPLY AND SERVICES CANADA
OTTAWA, CANADA

KILBORN

NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL A
WASTE MANAGEMENT AREA
DETAILS

5239 15

100-35-2 B



NOTE

1) ALL ELEVATIONS ARE IN METRES.

2) ALL DIMENSIONS ARE IN MILLIMETRES

3) ASSUMED MILL ELEVATION 65.0

100-35-4 DETAILS

REFERENCE DRAWINGS

B RELEASED FOR FINAL REPORT MAR 30 1984 R.W.

A RELEASED FOR DRAFT REPORT MAR 12 1984 E.N.

REVISIONS

CIVIL LAYOUT
1:2000
T.H.WALBAUM FEB 1984
P.A.FAHIE 6 FEB 1984

SUPPLY AND SERVICES CANADA
OTTAWA, CANADA

KILBORN

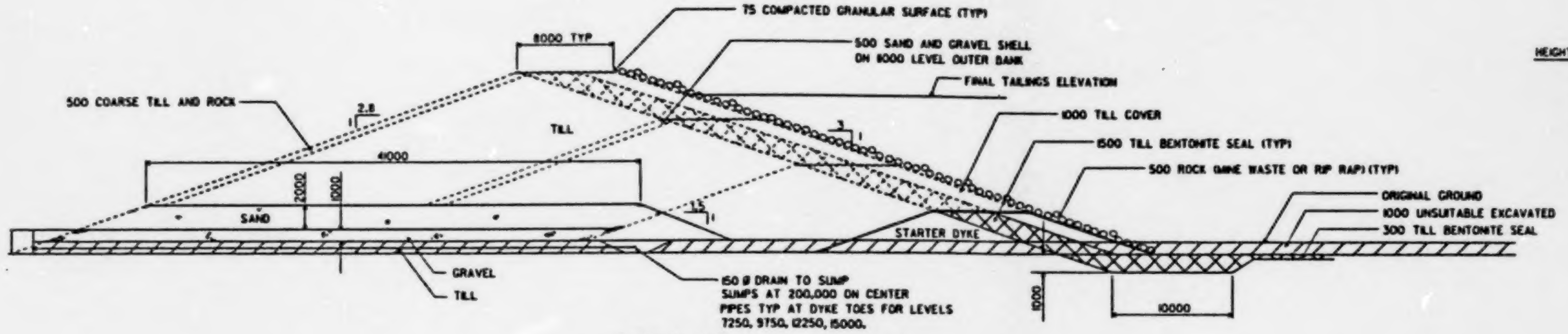
TITLE

NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY

MODEL B
WASTE MANAGEMENT AREA
SITE PLAN

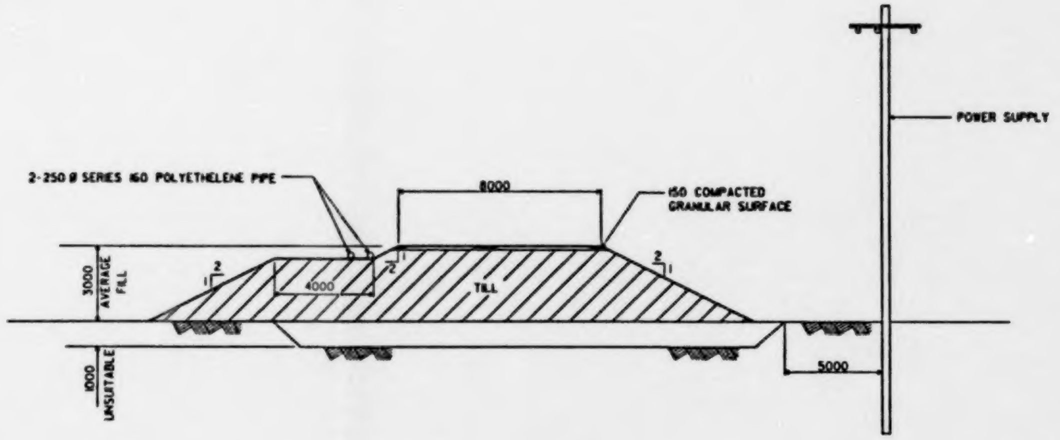
5239
15

100-35-3
B

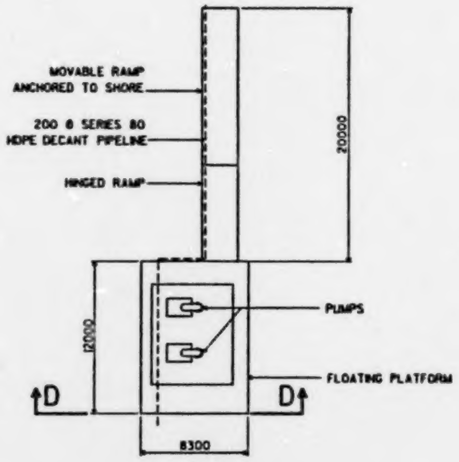

75 COMPACTED GRANULAR SURFACE (TYP)
8000 TYP
500 SAND AND GRAVEL SHELL ON 8000 LEVEL OUTER BANK
FINAL TAILINGS ELEVATION
500 COARSE TILL AND ROCK
2.8
1
TILL
3
1
1000 TILL COVER
1500 TILL BENTONITE SEAL (TYP)
41000
500 ROCK (MINE WASTE OR RIP RAP) (TYP)
2000
1000
SAND
1.5
1
STARTER DYKE
ORIGINAL GROUND
1000 UNSUITABLE EXCAVATED
300 TILL BENTONITE SEAL
GRAVEL
TILL
150 Ø DRAIN TO SUMP
SUMPS AT 200,000 ON CENTER
PIPES TYP AT DYKE TOES FOR LEVELS
7250, 9750, 12250, 15000.
1000
10000

SECTION A-A
(100-35-3)
SCALE 1:200

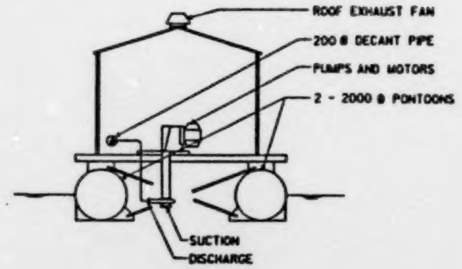
| HEIGHT OF DYKE | YEAR |
|---|---|
| 15000 | 18 |
| 13500 | 16 |
| 12250 | 14 |
| 11000 | 12 |
| 9750 | 10 |
| 8500 | 8 |
| 7250 | 6 |
| 6000 | 4 |
| 4750 | 2 |
| 3500 | 0 |


POWER SUPPLY
2-250 Ø SERIES 160 POLYETHELENE PIPE
8000
150 COMPACTED GRANULAR SURFACE
2
1
3000 AVERAGE FILL
4000
TILL
1000 UNSUITABLE
5000

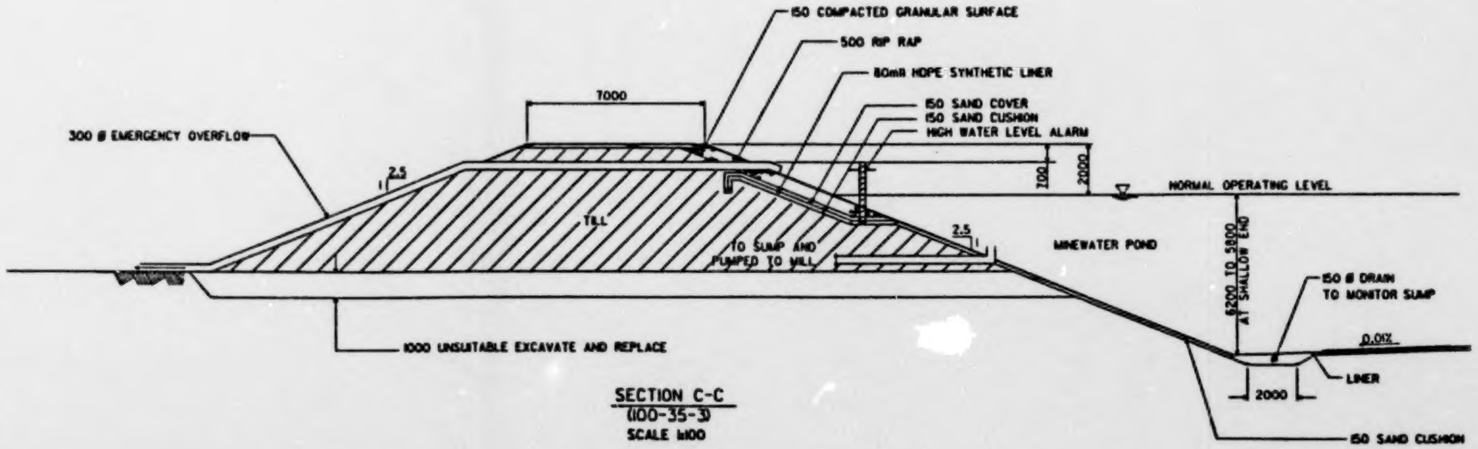
SECTION B-B
(100-35-3)
SCALE 1:100


MOVABLE RAMP ANCHORED TO SHORE
200 Ø SERIES 80 HDPE DECANT PIPELINE
HINGED RAMP
20000
12000
PUMPS
FLOATING PLATFORM
D
D
8300

BARGE - DETAIL 1
(100-35-3)
SCALE 1:200



SECTION D-D
SCALE 1:100



SECTION C-C
(100-35-3)
SCALE 1:100

SCALE 1:200
0 1000 5000 10000mm

SCALE 1:100
0 1000 5000mm

NOTE
ALL DIMENSIONS ARE IN MILLIMETRES

| | | | | |
|---|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR 30 1984 | R.W. | |
| A | RELEASED FOR DRAFT REPORT | MAR 17 1984 | E.N. | |
| | DESCRIPTION | | | |
| | REVISIONS | | | |

| | |
|---|---|
| 100-35-3 | SITE PLAN |
| | DESCRIPTION |
| | REFERENCE DRAWINGS |

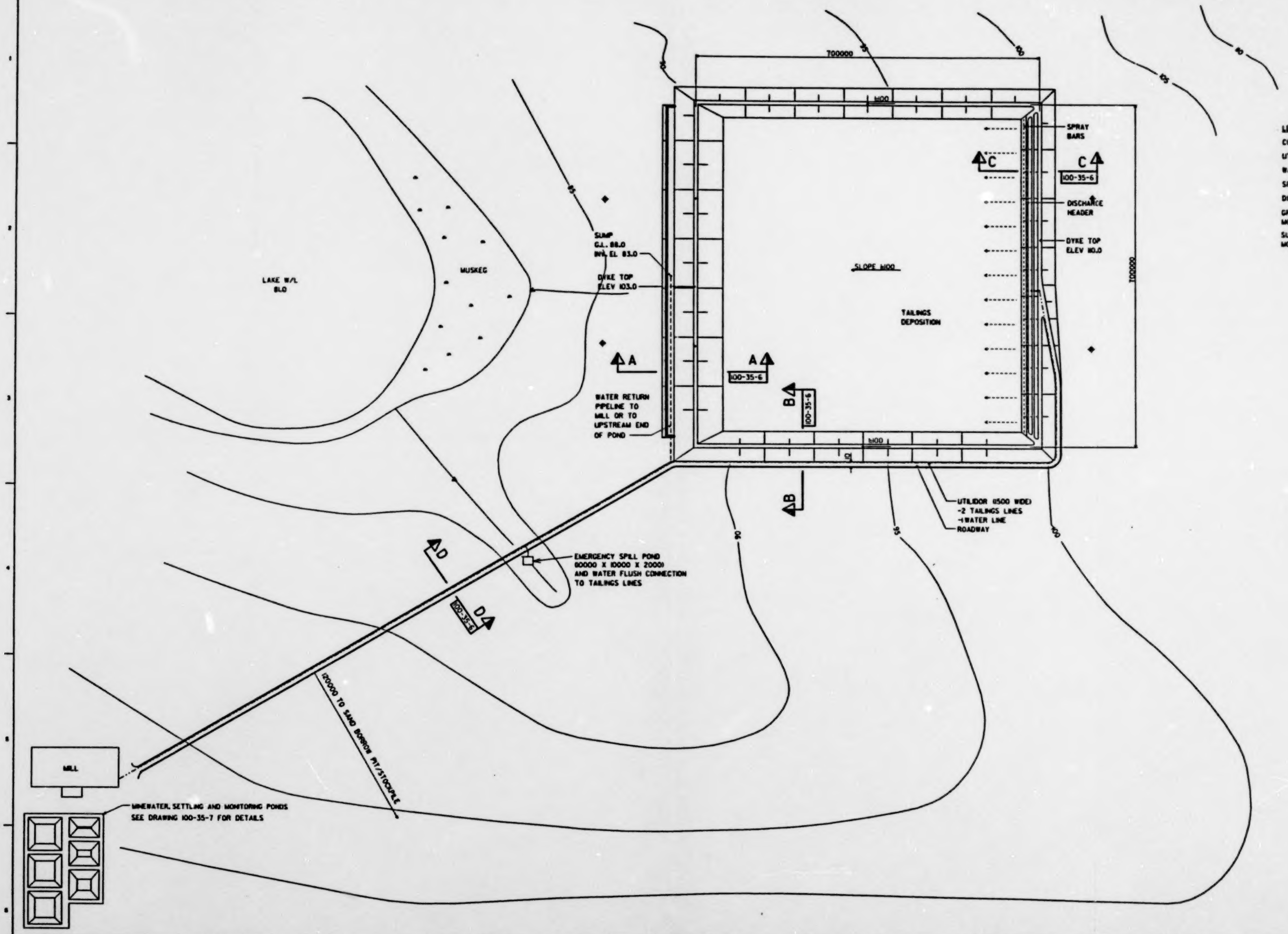
CIVIL LAYOUT
AS NOTED
T.N.WALBAUM
P.A.FAHE

SUPPLY AND SERVICES CANADA
OTTAWA, CANADA

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL B
WASTE MANAGEMENT AREA
DETAILS

S239 | 15
100-35-4 | B



**LEGEND**

| | |
|---|---|
| CONTOUR | 100 |
| UTILIDOR | —·—·—·— |
| WATER RETURN PIPELINE | — — — — |
| SPRAY BARS | - - - - - - |
| DISCHARGE HEADER | —·—·—·— |
| GROUNDWATER MONITORING PIEZOMETER | ◆ |
| SURFACE WATER MONITORING STATION | △ |

0 50000 100000 200000mm

SCALE: 1:300

**NOTE**

1) ALL ELEVATIONS ARE IN METRES
2) ALL DIMENSIONS ARE IN MILLIMETRES
3) ASSUMED MILL ELEVATION 100.0
4) CONCEPTUAL LAYOUTS DESIGNED
BY KNIGHT AND PIESOLD LTD.

| | | |
|---|---|---|
| 100-35-7 | MINEWATER, SETTLING AND MONITORING POND | |
| 100-35-6 | WASTE MANAGEMENT AREA DETAILS | |
| DWG. No. | DESCRIPTION | |
| REFERENCE DRAWINGS | | |

| No. | DESCRIPTION | DATE | BY |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR 30 1984 | R.H. |
| A | RELEASED FOR DRAFT REPORT | MAR 12 1984 | E.N. |

REVISIONS

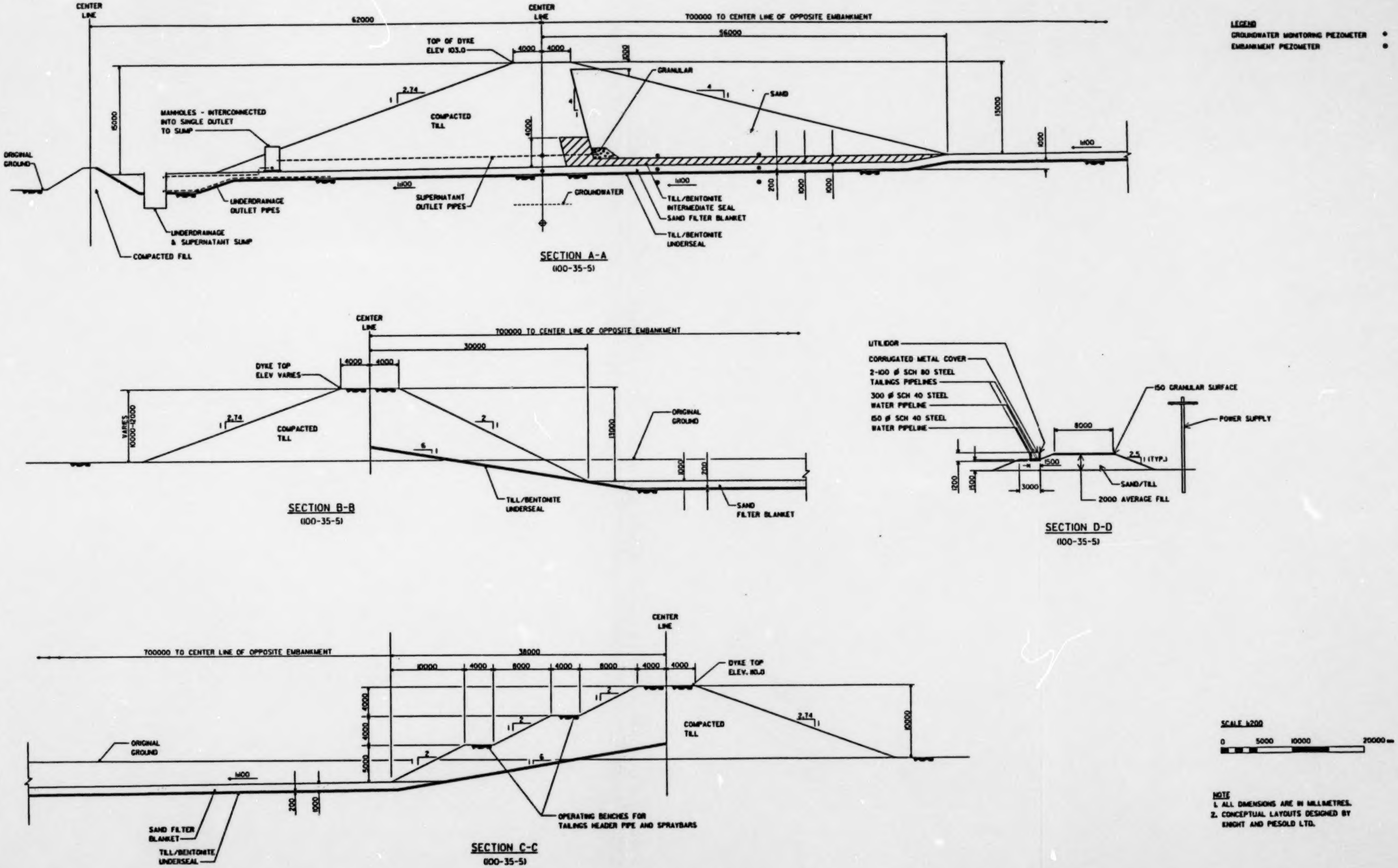
CIVIL LAYOUT
1:3000
J.P. HAILE FEB 1984
P.A.FAHIE 20 FEB 1984

SUPPLY AND SERVICES CANADA
OTTAWA, CANADA

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROJECT
WASTE MANAGEMENT COST STUDY
MODEL C
WASTE MANAGEMENT AREA
SITE PLAN

5239 15
DRAWING NUMBER
100-35-5 B



**LEGEND**

GROUNDWATER MONITORING PIEZOMETER ●

EMBANKMENT PIEZOMETER ●

**NOTE**

1. ALL DIMENSIONS ARE IN MILLIMETRES.
2. CONCEPTUAL LAYOUTS DESIGNED BY KNIGHT AND PIESOLD LTD.

| REFERENCE DRAWINGS | |
|---|---|
| 100-35-5 | SITE PLAN |

| REVISIONS | | |
|---|---|---|
| B | RELEASED FOR FINAL REPORT | RB |
| A | RELEASED FOR REPORT | CWC |

CIVIL LAYOUT
1:200
J.P.HAILE
P.A.FAHE

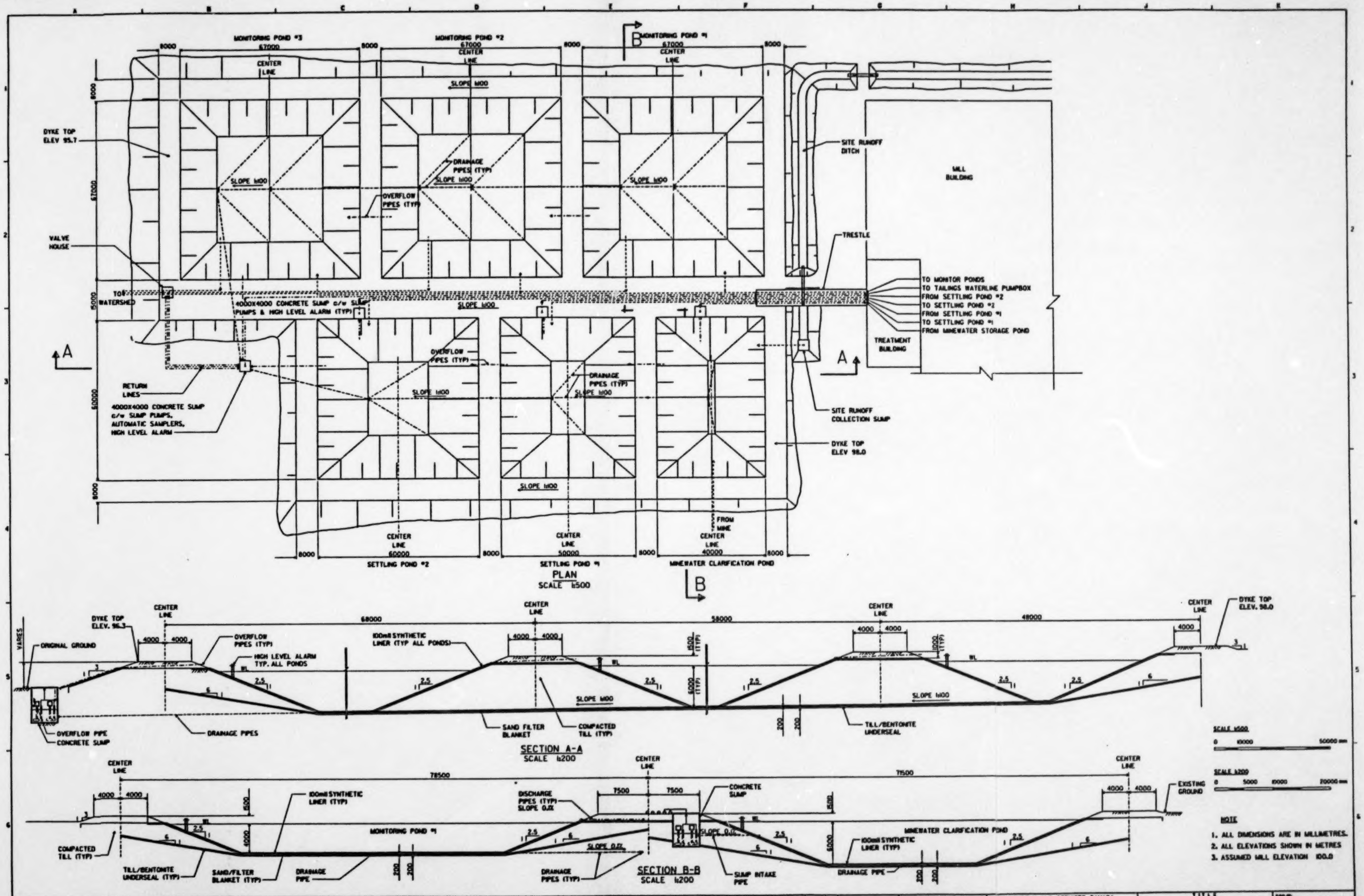
SUPPLY AND SERVICES CANADA
OTTAWA, CANADA

KILBORN

NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
WASTE MANAGEMENT AREA
DETAILS

5239 | 6

100-35-6 | B



NOTE

1. ALL DIMENSIONS ARE IN MILLIMETRES.
2. ALL ELEVATIONS SHOWN IN METRES
3. ASSUMED MILL ELEVATION 100.0

| | |
|---|---|
| 100-35-5 | SITE PLAN |
| DWG. NO. | DESCRIPTION |

REFERENCE DRAWINGS

| | | | |
|---|---|---|---|
| B | RELEASED FOR FINAL REPORT | MAR. 7/84 | ALL |
| A | RELEASED FOR DRAFT REPORT | MAR. 12 1984 | CMC |
| NO. | DESCRIPTION | DATE | BY |

REVISIONS

CIVIL LAYOUT
AS NOTED
T.H.WALBAUM FEB. 7 1984
P.A.FAHIE FEB. 10 1984

CLIENT: SUPPLY AND SERVICES CANADA
LOCATION: OTTAWA, CANADA

KILBORN

TITLE
NATIONAL URANIUM TAILINGS PROGRAM
WASTE MANAGEMENT COST STUDY
MODEL C
MINEWATER, SETTLING AND MONITORING POND
PLAN AND SECTIONS

PROJECT NO. 5239
DIVISION NO. 15
DRAWING NUMBER 100-35-7
REV. B

# APPENDIX A

# ORIGINAL STATEMENT OF WORK

File No: 26SQ.23241-3-1653
Serial No: OSQ83-00232

# APPENDIX A

## STATEMENT OF WORK

## COST ESTIMATE FOR CANADIAN URANIUM TAILINGS MANAGEMENT

The contractor will determine all costs of waste management at the uranium ore processing facilities as represented by each of Models A, B and C detailed in Appendix B. Capital and operating costs will be determined in the following areas:

1. Mine water treatment
2. Mill effluent treatment
3. Tailings line
4. Tailings storage basin
5. Tailings decant treatment
6. Monitoring

Capital Costs will include:

1. Design and engineering
2. Site preparation
3. Construction
4. Installation of equipment
5. Administration
6. Commissioning
7. Contingencies
8. Escalation

Operating Costs will include:

1. Chemicals and supplies including inventory costs
2. Labour and supervision

... 2

3. Services (i.e. steam, power)
4. Maintenance including warehouse costs
5. Depreciation

Excluded Costs are those derived from:

Base line environmental studies

Environmental hearings

General property facilities such as housing, airports, administration offices

Pumping mine water from mine to treatment facility

Costs and net revenue in recovering uranium in waste treatment

Decommissioning.

It is assumed that the facilities are in full production in late 1983, and will be decommissioned in 2003 to 2005. Capital and operating costs are to be determined in 1984 dollars. Operating costs are to be calculated on an annual basis, 350 operating days per year, and are to be determined per tonne of ore processed and per kg U recovered. The calendar year 1984, will be the fiscal year in which costs are to be determined.

Costs are to be determined from conceptual design layout without detailed engineering. Cost estimation should be within 30 percent accuracy. The contractor may need to obtain cost estimates from suppliers. In the absence of required design information in Models A, B and C, the contractor will assume conditions applicable. Conservative and best practical technology and engineering will apply. Because the waste management facilities must operate continuously, accepted practice of standby equipment and spares should be included.

# APPENDIX B

# ORIGINAL DESIGN BASIS

File No: 26SQ.23241-3-1653
Serial No: OSQ83-00232

# APPENDIX B

## BASIS FOR COST ESTIMATE

## CANADIAN URANIUM TAILINGS MANAGEMENT

Three conceptual situations for uranium tailings management are specified; one in Ontario, two in Saskatchewan. Tailings production results from milling ores at a nameplate production rate of $5x10^6$/lb $U_3O_8$ ($1.92x10^6$kg U) per annum. The production facilities are subject to the following conditions:

### I GENERAL CONDITIONS

| Model | A | B | C |
|---|---|---|---|
| Location | Ontario<br>Elliot Lake | Saskatchewan<br>Athabaska Basin | Saskatchewan<br>Athabaska Basin |
| Milling Rate tonnes/day | 6900 | 1420 | 440 |
| Ore grade % $U_3O_8$ | 0.10 | 0.475 | 1.5 |
| Mine t/d (5 days) | | | |
| Underground | 9660 | 1500 | - |
| Open Pit | - | 500 | 620 |
| Mine Water L/sec | 40 | 15-30 | 15-40 |
| Waste Management Type | Conventional wet/dry containment rock base | Wet/dry containment glacial till base | Layered deposition partially dry |

### II TIMING AND FINANCIAL ASSUMPTIONS (All Models)

Costs: dollars Canadian 1984

Engineering Start: January 1980

Construction Start: June 1981

Plant Start-up: August 1983

... 2

Operating Life: 20 years

Annual Inflation: 10%

Borrowing: 12.5%, engineering and capital expenditures spread evenly over pre-production period

Capital Depreciation: straight-line 10 years

## III LOCAL CONDITIONS

| Model | A | B | C |
|---|---|---|---|
| Services: Road: | all weather road to property gate maintained by Province | | |
| Power: | Ontario Hydro<br>diesel standby | diesel generated | diesel generated |
| Aircraft: | not required | personnel on 7-7 commuting | 7-7 commuting |
| Warehousing: | adequate spares and supplies maintained on company premises | | |
| Labour rates: | union | union | union |
| Analytical services: | company owned and operated at property | | |

## IV MILLING

### a) Ore Grade

| Model | A | B | C |
|---|---|---|---|
| | | % | |
| $U_3O_8$ | 0.10 | 0.475 | 1.5 |
| $Th^{232}$ | 0.02 | 0.001 | 0.01 |
| As | - | 0.02 | 1.0 |
| Ni | - | 0.02 | 1.5 |
| Fe | 3.0 | 2.5 | 3.0 |
| S | 3.5 | 0.3 | 0.8 |
| $CO_2$ | - | 2.0 | 1.0 |

... 3

| | A | B | C |
|---|---|---|---|
| b) Mineralogy | brannerite | pitchblende | pitchblende |
| | uranite | uranophane | gersdorffite |
| | pyrite | dolomite | millerite |
| | quartzite | quartzite | kaolinite |
| | | | chlorite |
| | | | quartzite |
| c) Ore Grind | 60%-250μm | 50%-74μm | 85%-250μm |
| d) Leaching | | | |
| acid | sulphuric | sulphuric | sulphuric |
| oxidant | air | chlorate | oxygen |
| vessels | pachucas | agitated tanks | autoclaves |
| temperature °C | 75 | 60 | 70 |
| acid consumption kg/t | 40 | 100 | 120 |
| leaching time hrs | 36 | 10 | 6 |
| e) Solids/Liquid Separation | | | |
| | filtration | C.C.D. | C.C.D. |
| stages | 2 | 6 | 8 |
| f) Uranium Recovery | strong base moving-bed ion exchange | solvent extraction | solvent extraction |
| | | strong acid strip | ammonia strip |
| | sulphuric acid strip | magnesia-hydrogen peroxide precipitation | ammonia precipitation |
| | LAMIX precipitation and neutralization | | ammonium sulphate recovery |

... 4

## V MINEWATER TREATMENT

### Model A

Composition: typical Elliot Lake

Treatment: air oxidation, lime, flocculant precipitation of U sludge to leaching, overflow to process

### Model B

Composition: pH6, 4 to 10° C, 5-15 mg/L $U_3O_8$
dissolved, 200 pCi/L $Ra^{226}$,
suspended solids 0.01 to 0.5 g/L

Treatment: settling of solids to process,
clear water to process water with overflow to tails

### Model C

Composition: pH6, 2 to 20°C, 5 mg/L $U_3O_8$,
50 mg/L As, 100 mg/L Ni,
1500 pCi/L $Ra^{226}$
suspended solids 0.01 to 0.5 g/L

Treatment:

(1) Settlement of solids to less than 5 mg/L in lined settling pond, minimum retention time 6 hours.

(2) Radium precipitation at pH8 (lime), 25 mg/L $BaCl_2$, flocculant. Settling in lined settling pond 10 hours minimum retention time. Filtration of pond decant with dual media sand filters, backwash to settling pond.

(3) Precipitation of metals with lime and ferric sulphate at pH 12. Lined settling pond with 48 hours retention time.

(4) Two parallel monitoring ponds with sulphuric acid neutralization to pH7. Release to environment. 4 days retention time each.

(5) Removal of sludge from each pond annually. Pond #1 sludge to mill neutral thickeners. Ponds #2 and 3 to tailings. Provision for 10 to 90% of flow from Pond #1 to be fed to process.

Ponds located adjacent to mill site. Separate treatment building. Provision to heat Pond #2 decant to 12°C with diesel cooling water (40°C).

... 5

## VI TAILINGS TREATMENT AND DISPOSAL - MODEL A

### 1. Tailings Neutralization

Tailings solids. IX barren and process water combined

3 stage air-agitated pachuas

6 hours retention time total

3.3 kg/tonne lime (to pH 10.5)

10 kg/tonne dolomitic lime

Discharge at 45 to 55% solids

### 2. Tailings Line

Length 2 km

Sclair pipe supported as necessary

Wet seal slurry pumps

Static hydraulic head 50m (steady uphill)

Standby line

4 hours capacity emergency-spill basin (rock) at feed end

Mass flow and spill alert instrumentation on lines

### 3. Tailings Basin

Complete construction before start-up

Displacing lake roughly rectangular shape, length-to-width 3:1
Drainage basin 4 times lake area

Twenty years capacity

50% of tails below original water line

Steep (25% slope) impervious rock on three sides

... 6

Typical eastern forest to be cleared below final tailings deposition level

Three dams to be constructed of glacial till with impervious clay core. Engineered for long-term stability.

Dimensions:

| Dam | Length(m) | Height, max (m) |
|---|---|---|
| 1 | 20 | 4.5 |
| 2 | 50 | 9.5 |
| 3 | 100 | 15.5 (above till) |

All dams to bedrock. Grouting of bedrock required. No excavation required for dams 1 and 2. Five meters swamp and till required excavated to bedrock at dam 3. Typical glacial valley contours to be assumed for all three dam sites.

Permanent road construction 5 km

Temporary road construction 2 km

Decant at dam 3

Till source 3 km haulage

4. Decant Treatment

$BaCl_2$ addition 25 mg/L

Two $BaSO_4$ settling ponds with 100 hours retention each. Pond #1 till-clay seal, Pond #2 synthetic liner. Sludge to be recovered at end-of-life for disposal underground. 25% of volume loss to sludge permitted to give 75 hours retention time at year 20.

Enclosed barium chloride mixing and distribution plant 100 hours mixed chemical storage. 90 days dry chemical

... 7

storage. Seasonal addition of flocculant. Five km 3 phase power line with substation. Ponds and mixing plant constructed on glacial till.

5. Monitoring

Sample frequency and constituent analyses as per AECB and MOE (Ontario) requirements.

All analyses completed in dedicated laboratory at mill site. 5-day operation.

Manpower and equipment as required.

Dams and basins instrumented for seepage.

## VII TAILINGS TREATMENT AND DISPOSAL - MODEL B

1. Tailings Neutralization

Tailings slurry, raffinate bleed and barren strip bleed treated to pH 8.5 with 30 kg/t slaked lime. 3 wooden agitated tanks, 2 hours retention time each.

Discharge at 45 to 55% solids.

2. Tailings Lines

2 Sclair lines on sand support, one operating, one spare. length 2 km.

Wet-seal pumps

Static hydraulic head 0

2 hours emergency-spill at feed end

Mass flow and spill alert instrumentation

3. Tailings Basin

Glacial till base

Minimum natural depression - assume flat

Site cleared by removing 1 meter of till

... 8

Engineered embankments constructed of compacted glacial till excavated at the site.

Engineered seal of till-bentonite in embankments and base

Maximum embankment height 20 meters

Top width 8 meters. Outside slope 2.8/1

Rectangular site, length/width 2:1

4. Decant Treatment

Floating decant pumping

Plant for precipitation of $Ba(Ra)SO_4$, flocculation, dual-media pressure filtration and return of sludge to tailings pond.

Remote control, operator attention 14 hours per week. Acid descale of media and equipment. Three phase power, propane fuel heat for building.

Tailings normally flooded. Surge capacity 7 days in tailings pond.

All operating equipment, except mix tanks, provided with standby.

Treated decant to watershed.

5. Monitoring

Sample frequency and constituent analyses as per AECB and Saskatchewan Environment requirements.

All laboratory analyses completed in mill laboratory, 7 day operation.

Manpower and equipment as required.

Tailings pond instrumented for seepage.

## VIII TAILINGS TREATMENT AND DISPOSAL - MODEL C

1. Tailings Neutralization

a) Raffinate

pH adjustment to 8 with lime in four air-agitated pachucas,

... 9

retention time 0.5 hours each. Sludge settled in a compact thickener and dewatered by vacuum filtration. Filtrate treated in a series of 4 mechanically-agitated tanks. $BaCl_2$ is added to tank #1, ferric sulphate and lime to bring pH to 11 in tanks 2 and 3. The overflow of tank #4 is settled in a compact thickener. The overflow is clarified in dual-media and sand filters and the clear liquid is acidified to pH 7 and discharged to synthetic-lined monitoring ponds for discharge to the watershed. The two parallel monitoring ponds have retention time of 48 hours each.

(Note: normally, minewater, after settling of suspended solids would be combined with raffinate for treatment. For purpose of this study, separate treatment is specified. Equipment is sized to include minewater).

Raffinate sludges are combined with slurry from CCD.

b) CCD Tails

The underflow from the final CCD thickener at 40% solids is treated with hydrated lime at the rate of 3 kg/t to bring pH to 11. Two agitated steel tanks in series with 4 hours retention time each are used. Recovered solids from the raffinate treatment circuit are added to tank #2.

Surge capacity of 72 hours is provided with 4 tanks of 18 hours capacity each, following the two tanks for lime addition. Tailings pumped at 35% solids and 30°C.

2. Tailings Line

Tailings pumps - variable-speed glandless diaphram, double stage

Tailings line-steel, duplicated, with expansion joints for -30 to +30°C

Utilidor - enclosed with one water pipe designed for 6000 $m^3$/day water discharge to the tailings containment area. Return seepage and runoff line designed for 3000 $m^3$/day

Tailings line length 2 km. Static head +10 m

Center depression with constant slope to -15 m

Emergency sump at center depression with 4 hours storage equipped with fresh water flush.

Utilidor constructed of fire retarding materials.

... 10

## 3. Tailings Storage

Designed for subareal method of deposition (Knight & Piesold design) with the following features:

- base excavated 2 m in glacial till
- underseal of till-bentonite mix sloping 1 in 100 toward mill complex; thickness minimum 200 mm, extending to center of embankments
- sand filter blanket 1 m thick with agricultural drain pipes
- confining embankments constructed of compacted till with locally available material
- all four embankments with 2.74:1 outside slope. Inside slope of three embankments 2:1. The downstream embankment will be zoned with the outside shell constructed of compacted till and the inside shell constructed of free-draining filter sand for collection of tailings runoff in an impervious trough at the base of the dam. Dam top width is 8 m for all four embankments.
- seepage and runoff are independently collected and monitored, then combined and pumped back to the mill to a lined storage pond of 7-day capacity, for continuous feed to the raffinate neutralization circuit.
- the total flow will be comprised of 100% of normal precipitation of rain and snow, and 20% of liquid contained in tailings as discharged
- mean annual precipitation is 460 mm per year
- the facility will be equipped with embankment piezometers and foundation piezometers to monitor flow conditions in the embankments and below the seal.

## 4. Monitoring

The neutralization plant and waste management facilities will be highly automated and instrumented. Sampling and analyses for internal and regulatory control will be extensive and will be performed on a 7-day basis at the site. All storage ponds will be instrumented for seepage and high level conditions.

# APPENDIX C

# MONITORING REQUIREMENTS

# TABLE C-1

## MODEL A
## MONITORING REQUIREMENTS

| Location | Parameters | Frequency |
|---|---|---|
| Decant Water | Flow | Daily |
| | pH, Acidity, Ra(tot) | Weekly |
| Treated Effluent | pH, TSS, Flow, TDS, Cu, Ni, Pb, Zn, Ra(diss) | Daily |
| | Alkalinity, Acidity, $NH_3$, $NO_3$, Ra(tot), Ra(diss) | Weekly |
| | TDS, $SO_4$, Cu, Ni, Pb, Zn, Mn, Mg, Fe, Co, $U_3O_8$, $PO_4$, Ca | Monthly |
| | TKN, $NO_2$, $NH_3$, $NO_3$, Cl, Na, K, Cu, Ba, Zn, Pb, Co, Ni, Mn, $SO_4$ | Annually |
| Minewater | Flow, $NH_3$, $NO_3$, pH, Acidity | Weekly |
| | Ni, Zn, Pb, Mn, Mg, TDS, Cu, Fe, Ra(diss) | Quarterly |
| Settling Pond Underdrains | pH, Acidity, Alkalinity, TDS, Flow, Ra(tot) | Monthly |
| Local Creek | pH, TDS, Ra(tot), Alkalinity, Acidity, Flow | Monthly |
| Downstream Station | pH, Fe, TDS, Alkalinity, Acidity, Flow, Ra(tot) | Monthly |
| | $SO_4$ | Quarterly |
| | TKN, $NO_2$, $NH_3$, $NO_3$, Cl, Na, K, Ca, Ba, Zn, Pb, $SO_4$, Co, Ni, Mn | Annually |

## TABLE C-1
## MODEL A
## MONITORING REQUIREMENTS (Cont'd)

| Location | Parameters | Frequency |
|---|---|---|
| Downstream Station | pH, Acidity, Alkalinity, Flow, TDS, Fe, Cu, Pb, Zn, Mg, Ni, Co, Mn, Ra(tot) | Monthly |
| | $SO_4$ | Quarterly |
| | TKN, $NO_2$, $NH_3$, $NO_3$, Cl, Na, K, Ca, Ba, Zn, Pb, Co, Ni, Mn, $SO_4$ | Annually |
| Downstream Station | pH, TDS, Ra(tot), Acidity, Alkalinity | Quarterly |
| | TKN, $NO_2$, $NH_3$, $NO_3$, Cl, Na, K, Ca, Ba, Zn, Pb, Co, Ni, Mn, $SO_4$ | Annually |

## TABLE C-2

## MODEL A
## ENVIRONMENTAL ANALYSES SUMMARY

| Parameter | Daily | Weekly | Monthly | Quarterly | Annually | Total/Year |
|---|---|---|---|---|---|---|
| pH | 1 | 2 | 4 | 1 | - | 521 |
| Acidity | - | 3 | 4 | 1 | - | 208 |
| Alkalinity | - | 1 | 4 | 1 | - | 104 |
| TSS | 1 | - | - | - | - | 365 |
| TDS | 1 | - | 5 | 2 | 4 | 437 |
| Ra(tot) | - | 2 | 4 | 1 | - | 156 |
| Ra(diss) | 1 | 1 | - | 1 | - | 421 |
| $U_3O_8$ | - | - | 1 | - | - | 12 |
| TKN | - | - | - | - | 4 | 4 |
| $NH_3$ | - | 2 | - | - | 4 | 108 |
| $NO_2$ | - | 2 | - | - | 4 | 108 |
| $NO_3$ | - | - | - | - | 4 | 4 |
| $SO_4$ | - | - | - | 2 | 4 | 12 |
| $PO_4$ | - | - | 1 | - | - | 12 |
| Cl | - | - | - | - | 4 | 4 |
| Na | - | - | - | - | 4 | 4 |
| K | - | - | - | - | 4 | 4 |
| Ba | - | - | - | - | 4 | 4 |
| Cu | 1 | - | 2 | 1 | - | 393 |
| Zn | 1 | - | 2 | 1 | 4 | 397 |
| Pb | 1 | - | 2 | 1 | 4 | 397 |
| Co | - | - | 2 | 1 | 4 | 32 |
| Ni | 1 | - | 2 | 1 | 4 | 397 |
| Mn | - | - | 2 | 1 | 4 | 32 |
| Fe | - | - | 3 | 1 | - | 40 |
| Mg | - | - | 2 | 1 | - | 28 |
| Ca | - | - | 1 | - | 4 | 16 |
| (Flow L/s) | 2 | 1 | - | - | - | 782 |
| TOTAL | 10 | 14 | 41 | 17 | 64 | 5,002 |

## TABLE C-3

## MODEL B
## MONITORING REQUIREMENTS

| Location | Parameters | Frequency |
|---|---|---|
| Sand Filter Discharge | Ra(diss), As, Cu, Pb, Ni, Zn, TSS | Daily |
| | Ra(tot), Ra(diss), pH Cond. | Weekly |
| | Temp., pH, Cond., Flow, $U_3O_8$, $SO_4$, Mg, $NH_3$, Ca, Pb, Ni, TSS | Bi-weekly |
| | TKN, $PO_4$, Cl, Ba, Na, K, $HCO_3$, $CO_3$ | Monthly |
| | Ether Sol., As, Cu, Zn, Mo, Se, V | Quarterly |
| | Ag, As, B, Ba, Cd, Co, Ci, Cu, F, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn, Pb 210, Th 230 | Semi-annually |
| Monitoring Wells (4 stations) | Temp., Cond., pH, TKN, Cl, $SO_4$, Ca, Na, Ra(tot) | Monthly |
| Toe Drains (3 stations) | Temp., Cond., pH, Flow, $U_3O_8$, TKN, $NH_3$, Cl, $SO_4$, Mg | Monthly |
| Groundwater Discharge (2 stations) | Temp., Cond., pH, $U_3O_8$, $SO_4$, Mg, Mn, TKN, Cl, Fe, Na, $PO_4$, $NO_3$, TSS, Ra(tot) | When Flowing |
| | Pb 210, As, Ba, Cd, Cr, Cu, Mo, Ni, Pb, Zn, Th 230 | Annually |
| | Ag, B, Co, F, Hg, Se, V | Annually |

TABLE C-3
MODEL B
MONITORING REQUIREMENTS (Cont'd)

| Location | Parameters | Frequency |
|---|---|---|
| Discharge Stream | Temp., pH, Cond., Ra(tot), $U_3O_8$, $SO_4$, Mg, Ca, $NH_3$, TKN | Bi-weekly |
| | K, Mn, Fe, Na, Cl, $CO_3$, $HCO_3$, TSS, Pb, $PO_4$ $NO_3$ | Monthly |
| | Pb 210, Th 230, As, Ba, Cd, Cr, Ca, Mo, Ni, Zn | Semi-annually |
| | Ag, B, Co, F, Hg, Se, V | Annually |
| Downstream Station | pH, Cond., Ra(tot), $U_3O_8$, TKN, $NH_3$, $SO_4$, Mg, TSS, Cl, $HCO_3$, Ca, K, Na, $CO_3$, $PO_4$, $NO_3$ | Quarterly |
| | Ag, As, B, Ba, Cd, Co, Cr, Cu, F, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn | Semi-annually |
| Downstream Station | Temp., pH, Cond., TSS, $U_3O_8$, Ra(tot), $SO_4$, Cl, $HCO_3$, $CO_3$, Ca, Na, Mg, K, tot. hardness, $PO_4$, TKN, $NH_3$, $NO_3$ | Quarterly |
| | Ag, As, B, Ba, Cd, Co, Cr, Cu, F, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn, Pb 210, Th 230 | Semi-annually |
| Lakes (7 stations) | pH, Cond., $SO_4$, TKN, $NO_3$ | Semi-annually |
| Drainage ditch (2) at Mine Area | Temp., pH, Cond., TSS, Fe, Mn, V, Zn, Ra(tot), $U_3O_8$ | Quarterly |

TABLE C-3
MODEL B
MONITORING REQUIREMENTS (Cont'd)

| Location | Parameters | Frequency |
|---|---|---|
| Drainage at Mine Area | Temp., pH, Cond., $U_3O_8$, TSS, Ra(tot), Ra(diss) | Monthly |
| | $PO_4$, TKN, $NH_3$, $NO_3$, $SO_4$, Cl, $HCO_3$, $CO_3$, Ca, Na, Mg, K, tot. hardness, Ag, As, B, Ba, Cd, Co, Cr, Cu, F, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn, Ra(tot), Pb 210, Th 230, $U_3O_8$ | Quarterly |
| Downstream from Mine Area | Temp., $U_3O_8$, TSS, Ra(tot) | Monthly |
| Inflow to Bay from Mine Area | Temp., pH, Cond., $U_3O_8$, TSS, Ra(tot) | Quarterly |
| Bay | pH, Cond., Ra(tot), Temp. | Quarterly |
| | Cu, Ni, Zn, Pb, TSS | Quarterly |
| | $PO_4$, TKN, $NH_3$, $NO_3$, $SO_4$, Cl, $HCO_3$, $CO_3$, Ca, Na, Mg, K, tot. hardness, Ag, As, B, Ba, Cd, Co, Cr, Cu, F, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn, Ra(tot), Pb 210, Th 230, $U_3O_8$ | Annually |
| Sediments (11 stations) | Ra(tot), Pb 210, Th(tot), $U_3O_8$ | Annually |
| Biota (3 stations) | Lake Trout, Northern Pike, Ra, Pb 210, $U_3O_8$, Th | Annually |

## TABLE C-4

## MODEL B
## ANALYSES SUMMARY - LIQUID

| Parameter | Daily | Weekly | Bi-Weekly | Monthly | Quarterly | Semi-Annual | Annually | Total/Year |
|---|---|---|---|---|---|---|---|---|
| Temp. | - | - | 2 | 9 | 5 | - | - | 180 |
| pH | - | 1 | 2 | 8 | 6 | 7 | - | 238 |
| Cond. | - | 1 | 2 | 8 | 6 | 7 | - | 238 |
| TSS | 1 | - | 1 | 3 | 3 | - | - | 465 |
| Ra(tot) | - | 1 | 1 | 6 | 7 | - | 1 | 179 |
| Ra(diss) | 1 | 1 | - | 1 | - | - | - | 429 |
| Pb 210 | - | - | - | - | 1 | 3 | 3 | 13 |
| Po 210 | - | - | - | - | - | - | 1 | 1 |
| Th 230 | - | - | - | - | 1 | 3 | 3 | 13 |
| $U_3O_8$ | - | - | 2 | 5 | 6 | - | 1 | 137 |
| $SO_4$ | - | - | 2 | 7 | 3 | 7 | 1 | 163 |
| $PO_4$ | - | - | - | 2 | - | - | - | 24 |
| TKN | - | - | 1 | 8 | 3 | 7 | 1 | 149 |
| $NO_3$ | - | - | - | 1 | 3 | 7 | 1 | 39 |
| $NH_3$ | - | - | 2 | 3 | 3 | - | 1 | 101 |
| $CO_3$ | - | - | - | 2 | 3 | - | 1 | 37 |
| $HCO_3$ | - | - | - | 2 | 3 | - | 1 | 37 |
| Na | - | - | - | 6 | 3 | - | 1 | 85 |
| Cl | - | - | - | 9 | 3 | - | 1 | 121 |
| Mg | - | - | 2 | 3 | 3 | - | 1 | 101 |
| Ca | - | - | 2 | 4 | 3 | - | 1 | 113 |
| K | - | - | - | 2 | 3 | - | 1 | 37 |
| Tot. Hard. | - | - | - | - | 2 | - | 1 | 9 |
| Ag | - | - | - | - | 1 | 3 | 4 | 14 |
| As | 1 | - | - | - | 2 | 4 | 3 | 384 |
| B | - | - | - | - | 1 | 3 | 4 | 14 |
| Ba | - | - | - | 1 | 1 | 4 | 3 | 27 |
| Cd | - | - | - | - | 1 | 4 | 3 | 15 |
| Co | - | - | - | - | 1 | 3 | 4 | 14 |
| Cr | - | - | - | - | 1 | 4 | 3 | 15 |
| Cu | 1 | - | - | - | 3 | 4 | 3 | 388 |
| F | - | - | - | - | 1 | 3 | 4 | 14 |
| Fe | - | - | - | 1 | 3 | 3 | - | 30 |
| Hg | - | - | - | - | 1 | 3 | 4 | 14 |
| Mn | - | - | - | 1 | 3 | 3 | 1 | 31 |
| Ni | 1 | - | 1 | - | 2 | 4 | 3 | 410 |
| Pb | 1 | - | 1 | 1 | 2 | 3 | 3 | 420 |
| Se | - | - | - | - | 2 | 3 | 4 | 18 |
| V | - | - | - | - | 4 | 3 | 4 | 26 |
| Mo | - | - | - | - | 2 | 4 | 3 | 19 |
| Zn | 1 | - | - | - | 5 | 4 | 3 | 396 |
| TOTAL | 7 | 4 | 21 | 93 | 106 | 103 | 77 | 5,158 |

## TABLE C-5

## MODEL B
## ANALYSES SUMMARY - SEDIMENT AND BIOTA

| | Annual |
|---|---|
| Ra | 14 |
| Pb 210 | 14 |
| Th | 14 |
| $U_3O_8$ | 14 |

## TABLE C-6

## MODEL C
## MONITORING REQUIREMENTS

| Location | Parameters | Frequency |
|---|---|---|
| Treated Effluent | As, Cu, Pb, Ni, Zn, TSS, Ra(tot) $U_3O_8$, pH, Temp., Cond. | Daily |
| | Ca, Mg, K, Na, $CO_3$, $HCO_3$, $SO_4$, Cl, TKN, TOC, $NH_3$, $PO_4$, TSS, Pb 210, Fe, Mn, Ba, Al, Po 210, Th, 230, Cd, Co, Cr, Hg, Mo, Se, V | Weekly |
| Seepage & Runoff | Temp., pH, Cond., As, Ca, Ni, Pb, Zn, Cd, Cr, Fe, Co, Hg, Mo, Se, V, Mn, Ba, Al, Ca, Mg, K, Na, $CO_3$, $HCO_3$, $SO_4$, Cl, TKN, $PO_4$, TSS, Ra(tot), $U_3O_8$, Po 210, Th 230, Pb 210 | Quarterly |
| Minewater Pond | pH, Cond., As, $SO_4$, Cl, Ra(tot), $U_3O_8$ | Quarterly |
| Tailings Area Piezometers (15) | pH, Conductivity, Temp., Ca, $SO_4$ | Monthly |
| | pH, Cond., Temp., TSS, Mg, K, Na, Cl, Ca, $SO_4$, P, TKN, As, Ni, Zn, Cr, Pb, Fe, Mn, Cu, Cd, Co, Hg, Al, Mo, Se, V, $U_3O_8$, Ra(tot), Pb 210 | Quarterly |
| Ponds Area Piezometers (5) | pH | Monthly |
| | $U_3O_8$, As, Ni, Ra(tot) | Quarterly |

### TABLE C-6
### MODEL C
### MONITORING REQUIREMENTS (Cont'd)

| Location | Parameters | Frequency |
|---|---|---|
| Ore Stockpile Seepage Collection | pH, Temp., Cond., $U_3O_8$, Ra(tot), As, Cu, Pb, Zn, Ca, Mg, K, Na, $CO_3$, $HCO_3$, $SO_4$, Cl, TKN, $PO_4$, TSS, Pb 210, Fe, Mn, Ba, Al, Po 210, Th, 230, Cd, Co, Cr, Hg, Mo, Se | Semi-annually |
| Dewatering Well | pH, Temp., Cond., $U_3O_8$, Ra(tot), As, Cu, Pb, Zn | Weekly |
| | Ca, Mg, K, Na, $CO_3$, $HCO_3$, $SO_4$, Cl, TKN, $PO_4$, TSS, Pb 210, Fe, Mn, Ba, Al, Ni | Monthly |
| | Po 210, Th 230, Cd, Co, Cr, Hg, Mo, Se | Semi-annually |
| Downstream (4 stations) | pH, Temp., Cond., Al, As, Ba, Cd, Co, Cr, Cu, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn, $U_3O_8$, Ra(tot), Pb 210, Po 210, Th 230, Ca, Mg, K, Na, $CO_3$, $HCO_3$, $SO_4$, Cl, $PO_4$, TKN, $NH_3$, TOC, TSS | Semi-annually |
| Stream Discharge | pH, Cond., Temp., $U_3O_8$, Ra(tot), Pb 210, Zn | Monthly |
| | Al, As, Ba, Cu, Fe, Mn, TSS, TKN, $PO_4$ | Quarterly |
| | Ca, Mg, Na, Cl, $CO_3$, $HCO_3$, $SO_4$, Cd, Co, Cr, Hg, Mo, Ni, Pb, Se, V, Po 210, Th 230 | Semi-annually |
| Downstream (5 stations) | pH, Temp., Cond., TSS, Ra(tot), Al, As, Ba, Cd, Co, Cr, Cu, Fe, Hg, Mn, Mo, Ni, Pb, Se, V, Zn, $U_3O_8$, Pb 210, Po 210, Th 230, Ca, Mg, K, Na, $CO_3$, $HCO_3$, $SO_4$, Cl, TKN, $NH_3$, $PO_4$, TOC | Monthly |

TABLE C-6
MODEL C
MONITORING REQUIREMENTS (Cont'd)

| Location | Parameters | Frequency |
|---|---|---|
| Downstream (4 stations) | pH, Cond., Alkalinity, $SO_4$, TKN, $NO_2$-$NO_3$ | Semi-annually |
| Biota | Macrophytes - Pb 210, Ra 226, Th, $U_3O_8$, As (4 stations) | Annually |
|  | Northern Pike, Lake Whitefish - Pb 210, Ra 226, Th, $U_3O_8$ (3 stations) | Annually |
| Sediments (4 stations) | $U_3O_8$, Ra 226, Pb 210, Th 230, As, Ni, Pb, LOI, Moistures, TKN, $TPO_4$, TOC, $SO_4$, Ca | Annually |
| Soils (5 stations) | pH, Moisture, As, Ni, $U_3O_8$, Ra 226, Pb 210, Th, $SO_4$ | Bi-annually |
| Plants (2 stations) | % ash, As, Ni, Pb 210, Po 210, Ra 226, Th, $U_3O_8$ | Bi-annually |

## TABLE C-7

## MODEL C
## ANALYSES SUMMARY - LIQUIDS

| Parameter | Daily | Weekly | Monthly | Quarterly | Semi-annual | Total/ Year |
|---|---|---|---|---|---|---|
| Temp. | 1 | 1 | 21 | 16 | 5 | 743 |
| pH | 1 | 1 | 26 | 17 | 9 | 815 |
| Cond. | 1 | 1 | 21 | 17 | 9 | 755 |
| TSS | 1 | 1 | 6 | 17 | 4 | 565 |
| Ra(tot) | 1 | 1 | 2 | 22 | 5 | 539 |
| Pb 210 | - | 1 | 7 | 16 | 5 | 210 |
| Po 210 | - | 1 | 1 | 15 | 7 | 138 |
| Th 230 | - | 1 | 1 | 15 | 8 | 138 |
| $U_3O_8$ | - | 1 | 1 | 22 | 5 | 162 |
| $SO_4$ | - | 1 | 21 | 17 | 9 | 390 |
| $PO_4$ | - | 1 | 6 | 17 | 5 | 202 |
| TKN | - | 1 | 6 | 17 | 9 | 210 |
| $NH_3$ | - | 1 | 5 | - |  | 60 |
| $NO_2$-$NO_3$ | - | - | - | - | 4 | 8 |
| $CO_3$ | - | 1 | 6 | 1 | 6 | 140 |
| $HCO_3$ | - | 1 | 6 | 1 | 6 | 140 |
| Na | - | 1 | 6 | 16 | 6 | 200 |
| Cl | - | 1 | 6 | 17 | 6 | 204 |
| Mg | - | 1 | 6 | 16 | 6 | 200 |
| Ca | - | 1 | 21 | 16 | 6 | 380 |
| K | - | 1 | 6 | 16 | 6 | 200 |
| As | 1 | 1 | 5 | 23 | 5 | 579 |
| Alkalinity | - | - | - | - | 4 | 8 |
| Ba | - | 1 | 6 | 2 | 5 | 142 |
| Cd | - | 1 | 5 | 16 | 7 | 190 |
| Co | - | 1 | 5 | 16 | 7 | 190 |
| Cr | - | 1 | 5 | 16 | 7 | 190 |
| Cu | 1 | 1 | 5 | 17 | 5 | 555 |
| Fe | - | 1 | 6 | 17 | 5 | 202 |
| Hg | - | 1 | 5 | 16 | 7 | 190 |
| Mn | - | 1 | 6 | 17 | 4 | 200 |
| Ni | 1 | - | 6 | 21 | 5 | 531 |
| Pb | 1 | 1 | 5 | 16 | 5 | 551 |
| Se | - | 1 | 5 | 16 | 7 | 190 |
| V | - | - | 5 | 16 | 5 | 134 |
| Al | - | 1 | 6 | 17 | 5 | 202 |
| Mo | - | 1 | 5 | 16 | 7 | 190 |
| Zn | 1 | 1 | 6 | 16 | 5 | 563 |
| TOTAL | 10 | 34 | 267 | 549 | 229 | 11,206 |

## TABLE C-8

## MODEL C
## ANALYSES SUMMARY - SEDIMENT AND BIOTA

| Parameter | Annual |
|---|---|
| $U_3O_8$ | 11 |
| Ra 226 | 11 |
| Pb 210 | 11 |
| Th 230 | 11 |
| As | 8 |
| Ni | 4 |
| Pb | 4 |
| LOI | 4 |
| Moisture | 4 |
| TKN | 4 |
| $PO_4$ | 4 |
| TOC | 4 |
| $SO_4$ | 4 |
| Ca | 4 |

## TABLE C-9

## MODEL C
## ANALYSES SUMMARY - SOILS AND PLANTS

| Parameter | Annual |
|---|---|
| pH | 5 |
| Moisture | 5 |
| % ash | 2 |
| As | 7 |
| Ni | 7 |
| Pb 210 | 7 |
| Ra 226 | 7 |
| Th | 7 |
| $U_3O_8$ | 7 |
| $SO_4$ | 5 |
| Po 210 | 2 |